minerals research contract report
une 1981

SELECTIVE PLACEMENT OF STRIP MINE OVERBURDEN IN MONTANA

V Summary Report

Contract H0262032
Prepared by:
Reclamation Research Unit
Montana Agricultural Experiment Station
Montana State University
Bozeman, MT. 59717

BUREAU OF MINES ★ UNITED STATES DEPARTMENT OF THE INTERIOR
Minerals Environmental Technology

Reference to specific brands, equipment, or trade names in this report is made to facilitate understanding and does not imply endorsement by either the Bureau of Mines or Montana State University.

50272-101

REPORT DOCUMENTATION PAGE	1. REPORT NO.	2.	3. Recipient's Accession No.

4. Title and Subtitle
Selective Placement Of Strip Mine Overburden In Montana
*V Summary Report

5. Report Date
Approved June, 1981

6.

7. Author(s) D.J. Dollhopf, J.D. Goering, C.J. Levine, B.J. Bauman, D.W. Hedberg and R.L. Hodder

8. Performing Organization Rept. No.

9. Performing Organization Name and Address
Montana Agricultural Experiment Station
Reclamation Research Program
Montana State University
Bozeman, MT 59717

10. Project/Task/Work Unit No.

11. Contract(C) or Grant(G) No.
(C) H0262032
(G)

12. Sponsoring Organization Name and Address
Office of the Assistant Director - Mining
U.S. Bureau of Mines
Department of Interior
Washington, D.C. 20241

13. Type of Report & Period Covered
Final
July, 1976 - June, 1981

14.

15. Supplementary Notes
* Four technical reports preceded this final report. They can be obtained from NTIS; PB81-103186, PB81-103178, PB81-103160, and PB81-103194.

16. Abstract (Limit: 200 words)
The objective of this five year study was to investigate the means of establishing non-polluted ground-water and root zone systems in areas where surface mining intercepts physiochemically unsuitable overburden materials.

Overburden may need to be sampled on a 60 m grid in order to delineate materials unsuitable for reclamation and to accurately implement selective handling operations. Unsuitable overburden materials were buried successfully in a dragline operation with a minimum of delays, but costs were increased 12 to 53%. After three years the buried material remains above the reestablished aquifer and below the root zone. Construction of a clay cap over the buried material may not be required in semiarid environments.

It was shown that some unsuitable overburden materials do not have to be special handled. When inhibitory material constituted <5% of the total overburden volume, such material was essentially not detectable in the resultant spoil pile due to mixing/dilution processes. When inhibitory material exceeded 15% of the overburden volume it was always found in the resultant spoil pile, generally with a smaller extent, indicating partial dilution.

At least a portion of the postmine aquifer had hydraulic conductivity rates lower than the premine system, indicating the reestablished aquifer may, in part, conduct ground water very slowly. Postmine ground-water chemical quality in spoil materials had poorer quality compared to premine ground water.

17. Document Analysis a. Descriptors

Reclamation, Overburden, Coal, Hydrology, Geology, Toxicity

b. Identifiers/Open-Ended Terms

Spoil mixing, Aquifer reestablishment, Drilling methods

c. COSATI Field/Group

18. Availability Statement Unlimited	19. Security Class (This Report)	21. No. of Pages
	20. Security Class (This Page)	22. Price

(See ANSI–Z39.18)

See Instructions on Reverse

OPTIONAL FORM 272 (4–77)
(Formerly NTIS–35)
Department of Commerce

FOREWORD

This report was prepared by the Montana Agricultural Experiment Station Reclamation Research Program, Montana State University in Bozeman under USBM Contract number HO262032. This contract was initiated under the Minerals Environmental Technology Program. It was administered under the technical direction of the Spokane Research Center with Gregory G. Miller, Lewis M. McNay, and Donald L. Donner as the technical project officers. Ms. Darlene F. Wilson was the contract administrator for the Bureau of Mines. This report is a summary of the work recently completed as part of this contract during the period July, 1976 to June, 1981. This report was submitted by the authors in May, 1981.

The authors wish to express their appreciation to M.A.E.S. staff Dr. F.F. Munshower and Mr. D.R. Neuman for the water analyses presented in this report. Former staff member, Mr. W.D. Hall, should be credited with initial work on this project associated with geology and hydrology. We gratefully acknowledge the equipment operators and engineers of the Long Construction Company, for their help in coordinating field demonstrations. Lastly, we extend our appreciation to staff of the Western Energy Company for their cooperation and suggestions during the term of this research project.

TABLE OF CONTENTS

Page

Table of Contents (continued)

LIST OF TABLES

LIST OF FIGURES

LIST OF APPENDICES

I

INTRODUCTION

Increased coal demand from western states has created concern that coal mining operations may adversely affect the long-term quality of the land resource. Concern stems from the possibility that overburden and coal seam aquifers will take on, to a certain extent, the chemical characteristics of its new flow medium composed of spoils (Moran et al., 1978). Overburden and interburden zones considered unsuitable for reclamation purposes due to chemical or physical characteristics are encountered in this region. Therefore, it is important that unsuitable overburden be isolated from the hydrological system, and similarly, such material should not be deposited within the plant root zone which may adversely affect revegetation efforts.

Selective handling of unsuitable overburden materials can be separated into two categories. The first, and most commonly practiced, is to segregate the same stratigraphic unit throughout the mine area. In Montana, this is being performed at two mine sites where interburden materials are being selectively placed below the future root zone, but above the pit base where an aquifer tends to reestablish. The second category is when discontinuous zones of unsuitable overburden materials are encountered and special handled. This is not being practiced in the western states even though most mine sites contain a greater volume of unsuitable material in overburden compared to the generally unsuitable interburden unit which is special handled at some mines. This situation has developed from an inability to accurately locate these unsuitable-discontinuous zones in overburden, and from the difficulty in implementing a mine plan where the overburden handling technique would be in a constant state of change as different shapes and volumes of unsuitable materials were intercepted. This report deals specifically with the second category of selective overburden handling and presents results on overburden characterizing methods, selective burial methods, associated costs, spoil mixing phenomena with a dragline, and aquifer quality as a function of mining.

The objective of this five year study was to investigate the means of establishing non-polluted ground-water and root zone systems in areas where surface mining intercepts physiochemically unsuitable overburden materials. The research area was located near Colstrip, Montana at a new mine site where borehole analysis suggested that excessive concentrations of several elements in the overburden would be intercepted by mining. The mine is operated by the Western Energy Company, a subsidiary of the Montana Power Company, and the site is referred to as Area B (Figure 1).

This is the fifth and final technical report of a series on this project and all research results are summarized herein.

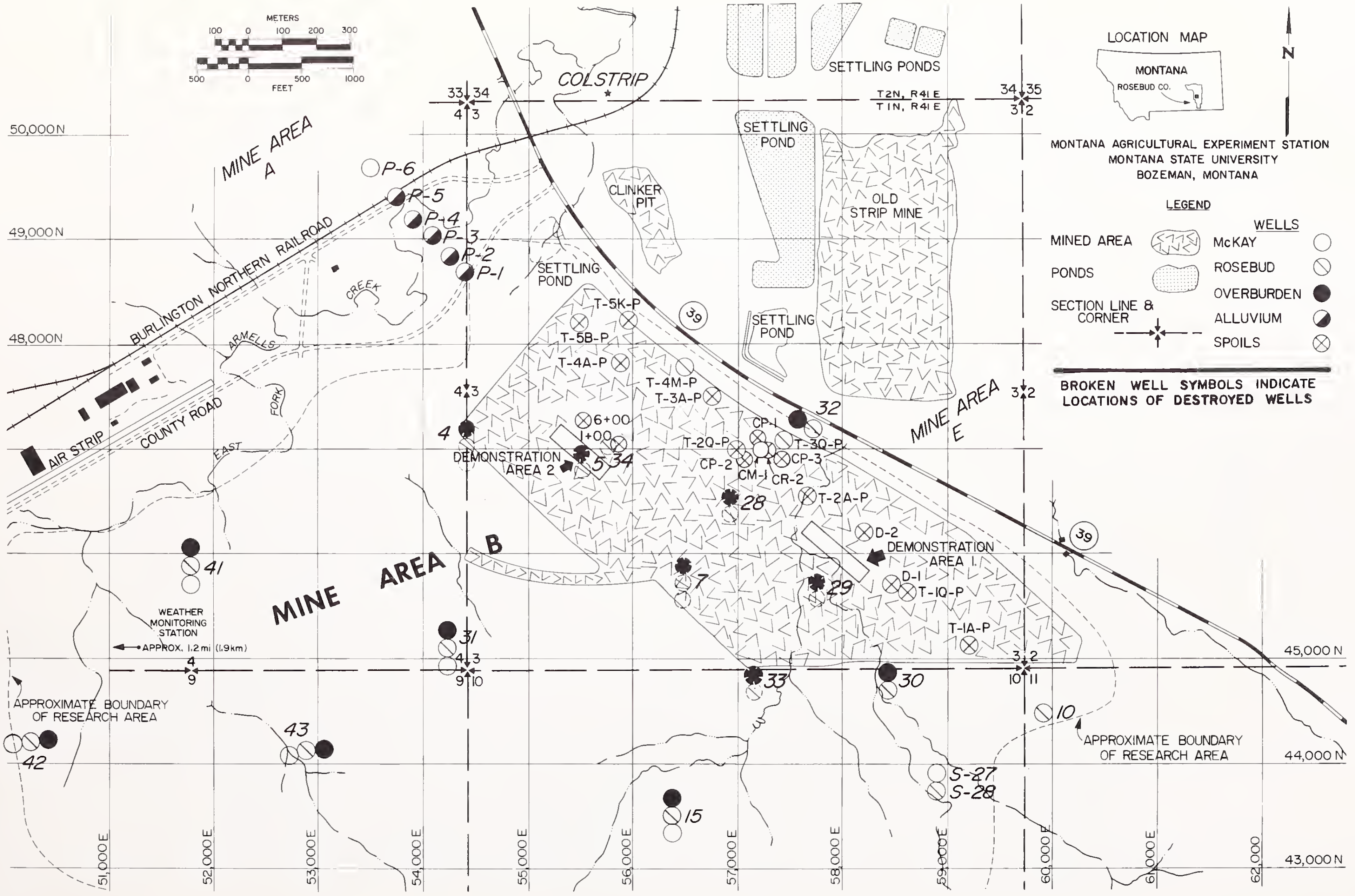


Figure 1. Location map of the research site at the Western Energy Company Mine Area B, near Colstrip, Montana.

II

RESEARCH AREA DESCRIPTION

PHYSIOGRAPHY

The research area is located approximately 1 km southwest of the town of Colstrip, Montana (Figure 1). The area topography, developed on nearly horizontal strata of the Tongue River Member of the Fort Union Formation (Paleocene), is typified by rolling hills and uplands developed on fine grained sediments and steep sloped plateaus capped with erosion resistant clinker and cemented channel sandstones (Geology section). The Rosebud and McKay Coals and to a lesser extent, sandstone in the overburden form important aquifers in the area. Regional uplift has rejuvenated streams and produced a dissected topography in many areas. Elevations range from 985 m (3240 ft) mean sea level (msl) at the East Fork Armells Creek to about 1106 m (3630 ft) msl in headwater areas. All streams in the research area are ephemeral except the East Fork Armells Creek, which is intermittent (Hydrology section). The research area is characterized by a typical continental semiarid climatic regime with a mean annual precipitation of approximately 33 cm recorded during the study period (Climatology section).

SOILS

Soils of the Colstrip vicinity lie in a transition zone of Aridisols, Alfisols, and Mollisols. Aridisols or desert soils are dominant in the Southern Powder River Basin in a somewhat drier climate. Mollisols, prairie soils, are common in the moister Great Plains of North and South Dakota. Alfisols or forest soils occur in the neighboring Big Horn mountains and Black Hills regions. Entisols which are young, undeveloped soils are locally common due to steep terrain. Field research at Colstrip indicates that the soil landscape is comprised primarily of Ustic Torriorthents, Borollic Camborthids, Aridic Haploborolls, Ustollic Haplargids, and Ustic Torrifluvents. A soils map of the research area as well as physiochemical characteristics of the soil types were presented in an earlier report (Dollhopf et al., 1978a).

Soil Forming Factors

Soils in the Colstrip area result from the interaction of several soil-forming factors. Climate, organisms, parent material, relief, and time have had an effect in determining the properties of soils in this region. These soil properties in turn also affect vegetation, hydrology, and land-use patterns.

As a result of low precipitation, runoff during snowmelt and high intensity thunderstorms, and high evapotranspiration rates, soils in the Colstrip area are dry in the root zone during much of the growing season. Many coarse-textured soils are dry long enough to have an aridic moisture regime which is typical of desert soils. Finer-textured soils with more clay and organic matter retain moisture longer into the summer and have a ustic moisture regime. Mean annual soil temperature is such that soils at low elevations have a mesic temperature regime while soils in the uplands have frigid temperature regimes (mean annual soil temperature less 8°C, 47°F). These facts suggest that climate can play an important part in the classification of soils in the Colstrip area.

Vegetation is another important soil-forming factor. The region is dominated by mixed grass prairie with small areas of Ponderosa pine uplands. Soils under grassland tend to build up thick A horizons high in organic matter due to the contribution of grass root systems (Mollisols). Soils under forest usually have an O horizon of relatively undecomposed organic matter with an underlying mineral A horizon low in pH. Argillic B horizons (clay accumulation) are also common in forest soils (Alfisols).

Most soils are dominated by a coarse-loamy particle-size in the Colstrip area. Fine-loamy soils are found where residuum from sandstone and siltstone are mixed. Fine-silty soils occur on siltstones. Some fine soils (more than 35% clay) occur on shale in alluvial valleys and other topographic lows. Skeletal and fragmental soils are common on top of clinker (porcelanite) buttes.

Relief is another soil-forming factor which plays an important rôle in determining the properties of soils in the Colstrip area. The relief factor includes the effect of both slope and aspect on soils. Due to the sufficiently steep gradient of streams draining the region, and the weakly consolidated nature of the Fort Union materials, a moderately steep dissected topography has resulted.

Differences in aspect can have a strong effect on microclimate which in turn affects soil development through vegetation differences. Soil on north-facing slopes tend to be colder and wetter because less incident energy is available to warm and dry these soils. Organic matter tends to accumulate to higher levels under these conditions. Soils on south-facing slopes are lower in organic matter.

The rate of geologic erosion is high on steep slopes in the uplands. As a result these soils cannot accumulate thick A horizons as the soil surface is slowly deflated. Steep surfaces are not stable long enough for cambic B horizons (horizon of alteration) to form. The resulting soils are classified as Entisols. On the midslopes, erosion is less and surfaces are more stable. Cambic B horizons form and slightly thicker A horizons accumulate. The dominant soils are Borollic Camborthids. On the lowest part of the slopes, gradients are shallow. Material that may be carried by overland flow from upslope is deposited here due to a loss in energy of the moving water. This material is often high in organic matter if it derived from A horizons of upland soils. Over-thickened mollic epipedons can result (A horizons high in organic matter). Due to the accumulation and retention of surface runoff in these geomorphic positions, more water moves through these profiles. Thus the solum (A and B horizon) is the thickest in these locations. If enough clay is present in the deposited material or in the residual parent material, argillic horizons may also form. Aridic Haploborolls and Borollic Haplargids dominate on the lower slopes. In the river channels and flood plains, soils are constantly reworked. Normal pedologic processes are often disturbed as material is deposited or removed by alluvial action. These soils with no B horizon are classified as Fluvents.

Time is the last important soil-forming factor. The Colstrip region is within the unglaciated portion of the Missouri Plateau so there have been no major surface disturbances in the area in geologically recent times. However, the downcutting of the rivers, which is evidenced by the numerous steep-sided coulees in the area, keeps the soil surface rejuvenated. Absolute dating of soils is extremely difficult so soils are termed "young" or "old" according to their profile development. Pronounced horizonation indicates an "old" soil. In the Colstrip area, young soils (Entisols) are common in the least stable upper slopes. All other soils in the area have medium development.

VEGETATION

Colstrip Area

Most native vegetation of the Colstrip area of southeastern Montana can be characterized as within either mixed prairie or coniferous woodland plant association, although scattered inclusions of riparian vegetation do occur along more mesic to hydric drainages and streamcourses. Payne (1973) classified the area in general as within the eastern Montana ponderosa pine savannah vegetative type, which contains scattered open stands of ponderosa pine interspersed among broad expanses of northern mixed prairie vegetation. Vegetation of the Colstrip area has been described at length previously (e.g. Sindelar et al., 1973, 1974; Meyn, Sundberg and Young, 1976; Munshower, Sindelar and Neuman, 1975; Munshower and DePuit, 1976; Lewis and Lefohn, 1976; Lewis, Glass and Lefohn, 1976).

Mixed prairie is the most prevalent class of native vegetation in the Colstrip area, and is composed of a diverse mixture of cool and warm season perennial grass, forb and shrub species. Mixed prairie plant communities are typically encountered on dry, level to steeply sloping uplands. Plant species composition, cover and production of specific sites varies markedly depending on soils, topography and microclimate (i.e., range site) and pattern-history of land use.

In the Colstrip region, relatively undisturbed sites are the exception rather than the rule since much of the area's rangeland has been rather continuously and sometimes excessively grazed by livestock. Overgrazing generally tends to promote development of various increaser or invader native shrub, half-shrub and weed forb species as well as certain introduced annual grasses and weeds. Hence, much native mixed prairie has retrogressed considerably from climax, being presently composed of a variable admixture of climax-decreaser, increaser, and invader plant species. Listing of common grass, forb, and shrub species of mixed prairie plant communities of the Colstrip area have been presented in other reports (e.g. Munshower et al., 1975; Munshower and DePuit, 1976; Lauenroth et al., 1976).

Coniferous woodland vegetation is scattered and rather sporadic in distribution in the Colstrip area, in general occupying sites of higher elevation, high relief, rough topography and/or northerly exposure. Ponderosa pine usually dominates, growing in open to moderately dense stands, although on certain sites Rocky Mountain juniper may be conspicuous. Understory vegetation consists of a variety of mixed prairie grasses, shrubs, and forbs as well as occasionally, more mesophytic broad-leaved shrubs. Riparian vegetation is encountered on sites with relatively greater available moisture, such as along stream courses, drainages or on subirrigated areas. If sufficient moisture is available, plant communities may be dominated by broad-leaved trees. On other sites, dominance may be exhibited by various broad-leaved shrubs, or smaller trees. Although rather sporadic and localized in distribution, this type of vegetation is nonetheless a conspicuous feature of the Colstrip area.

Livestock grazing constitutes the prime current land use of most areas of native vegetation of the Colstrip region. Payne (1973) assigned a mean estimated livestock carrying capacity for the eastern Montana ponderosa pine savannah vegetative type of 4.0 acres (1.6 ha) per animal unit month (A/AUM). Locally, carrying capacities may be greater or less than this level depending on range site and condition. Carrying capacities of higher condition mixed prairie range sites of the Colstrip area were noted between 1.5 and 3.0 A/AUM by Munshower and DePuit (1976), while lower condition or less productive sites were as low as 13.2 A/AUM. Total above ground plant productivity also varies widely according to range site and condition. In general, mixed prairie sites produce between 1000 and 1800 Kg/ha/year, with mean production levels in the range 1200-1300 Kg/ha/year (Munshower and DePuit, 1976; Lauenroth et al., 1976). Aboveground production by non-tree species with coniferous woodland plant communities may range roughly between 700 and 900 Kg/ha/year (Munshower and DePuit, 1976; DePuit et al., 1975).

A limited conversion of native rangeland into agricultural cropland has occurred in the Colstrip area, although the total acreage of agricultural lands is rather small relative to acreage of native vegetation. Principal agricultural land uses include hay-pasture and cereal grain production. Irrigated or dryland pastures are utilized as livestock pasturage or as sources of hay. Barley and winter wheat comprise the major cereal grain crops of the area, with typical average yields of 60 and 47 bushels per acre (.4 ha) respectively (Meyn, et al., 1976).

Study Area

Baseline (pre-mining) vegetation characterization of the study area was accomplished by a private consulting firm in 1975 (E.C.S., 1975); data of this characterization were synthesized and presented by Dollhopf et al. (1978a). Three broad categories of natural plant communities were found within the study area portion of Western Energy Company mining Area B. Approximately 21.5 percent (939 ha) of the area was collectively covered by five different grass dominated vegetation types, generally in fair to good range condition. Five shrubland-grassland vegetation types collectively comprised the largest portion of the area - 2568 ha, or 58.85 percent of the area. Condition of shrub-grassland vegetation types varied widely from poor to good depending on degree and history of grazing use. Timber-grassland communities covered only 1.8 percent (80.2 ha) of the study area.

These natural vegetation types occurred over a variety of range sites, ranging from overflow to thin-hilly. Excluding all non-natural vegetation classification units, silty appeared to be the most widespread range site, covering approximately 57.4 percent of the area, followed by clayey (14.0 percent), overflow (13.5 percent), sandy (12.0 percent), dense clay (2.5

percent) and thin-hilly (0.7 percent). Roughly 3.2 percent of the range sites were in poor range condition, 65.2 percent in fair condition, 13.3 percent in good condition and only 0.4 percent in excellent condition. The S.C.S. recommended livestock carrying capacity for natural vegetation types of the study area varied from 1.7 to 10 A/AUM. Total vegetation productivity ranged from 171.1 to 2324.0 Kg/ha. Details on floristic composition, vegetation structure and composition may be found in Dollhopf et al. (1978a).

Agricultural land and disturbed (i.e., cultivated) grassland together covered approximately 17.1 percent (749 ha) of the study area. Less than 1.0 percent of the area consisted of nonvegetated sandstone outcrops or cultural features (residential, stock ponds, etc.).

The range resource thus comprised the major land resource of the study area portion of Mine Area B, and although mainly in only fair condition due to livestock overgrazing, this resource still had definite value in terms of local livestock production. It would appear that reestablishment of suitable rangeland vegetation capable of withstanding and supporting grazing livestock will be the ultimate reclamation goal for most of this area according to present State of Montana regulations. The presence of a limited acreage of agricultural land within the pre-mining area, however, may complicate reclamation goals somewhat if exact restoration of pre-mining conditions is desired.

GEOLOGY

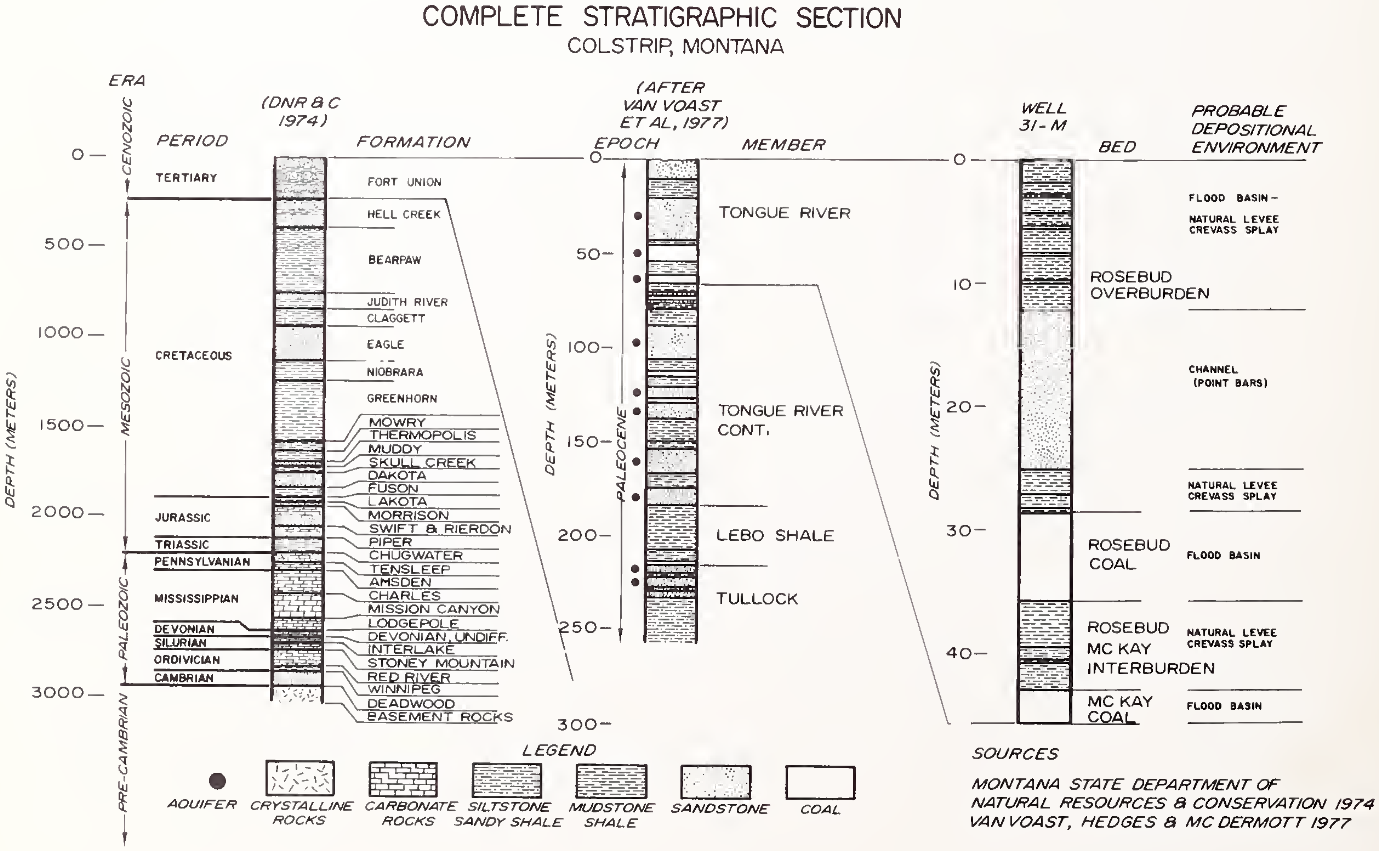
History

Much of southeastern Montana experienced extended periods of relative quiescence throughout most of the Paleozoic and Mesozoic Eras. Sediments were typically deposited in shallow marine epicontinental seas. Multiple transgressions and regressions of the sea produced strata of sandstone, shale and limestone. Although periods of gentle regional uplift and associated erosion created some gaps in the geologic record, little deformation occurred during this interval. Near the end of the Mesozoic Era, at the end of the Cretaceous, about 63 my ago, broad regional uplift and the initial effects of the Laramide Orogeny resulted in the withdrawal of the seas and initiation of continental deposition. Tectonic events continued into the Paleocene Epoch, during which time the Fort Union Formation was deposited over large areas of eastern Montana, Wyoming and Colorado. Large river systems flowed south and east from western highlands to the Cannonball Sea in what is now the Dakotas (Shurr, 1972; Royse, 1970). Tectonic events during the Eocene Epoch created most present-day geologic structures found in southeastern Montana. Volcanic activity to the west and orogenic sediments are thought to have buried most of the area with debris during the Oligocene and Miocene Epochs (Glaze and Keller, 1965). Major regional uplift near the end of the Pliocene rejuvenated streams, which, in turn, reexcavated basins and exhumed former upland areas. Erosion since the Pliocene has produced the present landscape.

Stratigraphy

The Tongue River Member of the Fort Union Formation is the only consolidated geologic unit exposed in the Colstrip vicinity. The unit is composed of interbedded massive sandstone, shale, siltstone, coal and clinker. The interval studied during this project included all materials above the base of the McKay Coal (Figure 2). The strata were divided into four units: the Rosebud overburden (5-65 m), the Rosebud Coal (5.5-7.9 m), the Rosebud-McKay interburden (3-9 m) and the McKay Coal (2-3 m).

The Rosebud overburden is characterized by interbedded massive sandstone units, siltstone, mudstones and shale. This unit is described more completely in the depositional model section (Chapter III). The Rosebud Coal generally averages 7 to 8 m thick in the research area. Where this seam has not been subjected to erosion or burning, it is widespread in the Colstrip vicinity and is mined by Western Energy Company (Rosebud Mine) and Peabody Coal Company (Big Sky Mine). The Rosebud-McKay interburden generally consists of gray silty shale with lenses of fine sand. Several miles to the west of the research site, this unit has been reported to thicken to 35 m (Chadwick et al., 1975). The McKay Coal is also laterally continuous over a wide area in the Colstrip vicinity, but has a higher sulfur content than the Rosebud Coal (Chadwick et al., 1975) and is not mined by the Western Energy Company in the Colstrip area.



COMPLETE STRATIGRAPHIC SECTION
COLSTRIP, MONTANA

SOURCES
MONTANA STATE DEPARTMENT OF NATURAL RESOURCES & CONSERVATION 1974
VAN VOAST, HEDGES & MC DERMOTT 1977

Figure 2. Complete stratigraphic section from the Western Energy Company Mine Area B, Colstrip, Montana (from Dollhopf, et al., 1978).

Structure

The Colstrip area lies near the northwestern boundary of the Powder River Basin. Laramide tectonic events created the basin between the Bighorn and Pryor Mountains to the west, the Black Hills to the east, the Miles City arch and Porcupine Dome to the north and the Laramie Mountains and Hartville uplift to the south. The southeast plunging Ashland syncline, located between the Bighorn Mountains and Porcupine Dome, dominates the research area. The axis of a broad, shallow syncline passes immediately southwest of the research area and plunges southeast at 1 to 3 degrees. Drilling log data indicate that numerous small scale folds also exist in the research area (Dollhopf et al., 1977b). These structures also generally plunge southeast. Some discordant small scale folds may be the result of differental sediment compaction. Carmichael (1964) inferred the existence of several faults in Area B. Although no conclusive evidence was found for them, anomalous well yields at certain locations and alignment of drainages strongly suggested some minor faulting has occurred.

III

HYDROLOGY AND WATER QUALITY

SURFACE HYDROLOGY

Climatology

Data from a weather station erected in January 1977 were gathered on precipitation, temperature, windspeed and direction, relative humidity, evaporation and solar radiation. Selected data are presented in the following paragraphs. A comprehensive data summary is available upon request.

Mean annual precipitation for the Colstrip area has been reported at 42.9 cm and 40.1 cm for the periods 1965-1974 (Toy and Munson, 1978) and 1941-1970 (U.S. Dept. of Commerce, NOAA), respectively. Data for the period January 1977 through December 1980 from the research site weather station indicated a mean annual precipitation of 32.81 cm. An average of 20.33 cm or 62% fell during the growing season (May through September) (Figure 3). Toy and Munson (1978) reported an average of 23.4 cm fell during the growing season for the period 1965-1974.

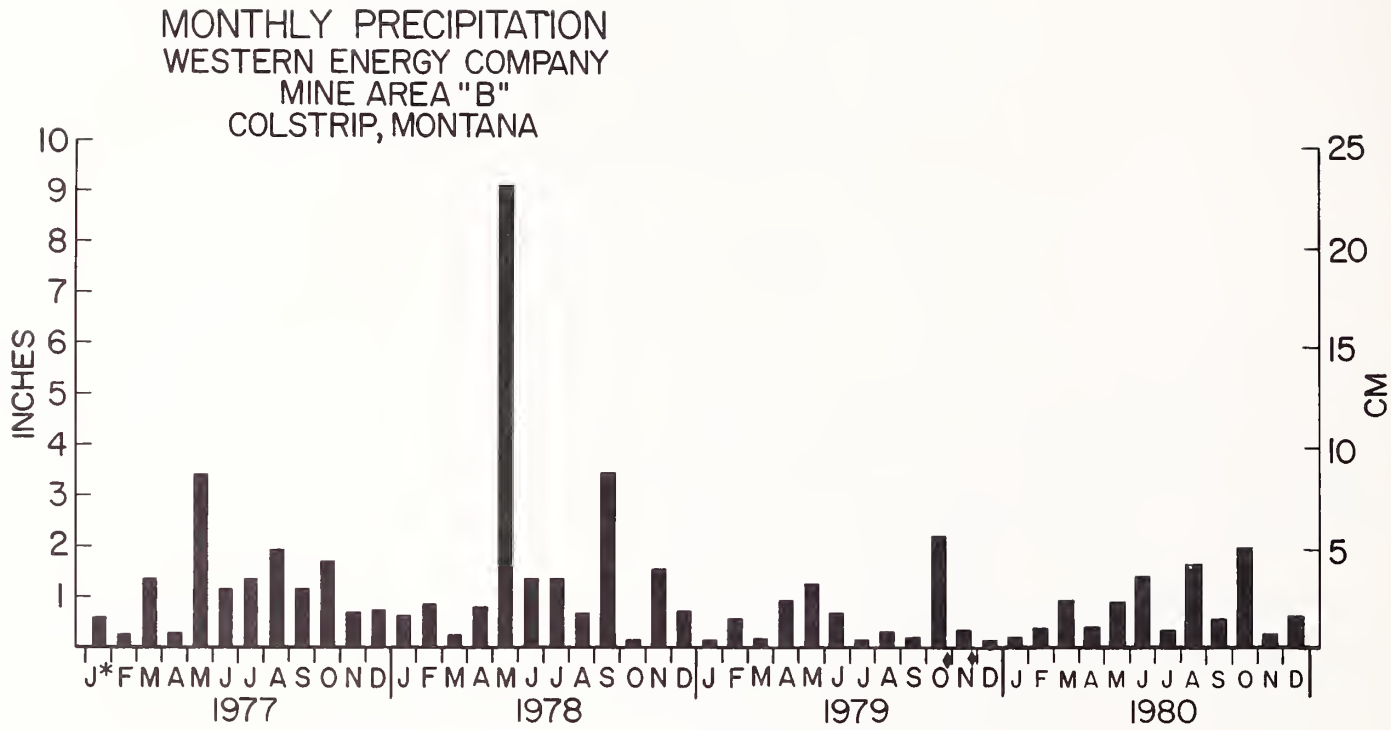
Winter snow cover is generally light. Windblown accumulations, however, in valleys and on lee slopes may be several meters in depth, likely inducing local infiltration and percolation. The area is characterized by dry cool winters, moderate precipitation during May and June, light to moderate precip-during July through October (primarily from thunderstorms) and a return to dry conditions in November. From Figure 3, it is apparent that much variation occurs from one year to the next, a situation typically encountered in most semiarid regions.

Streams and Ponds

The surface hydrology of the research area is dominated by several features: East Fork Armells Creek, located along the northern edge of the site, provides the main surface drainage; the settling pond across Highway 39, about N 48,000; E 57,300; and the open Area B mine pit (Figure 1).

Flow in the East Fork Armells Creek was observed only during spring snowmelt or prolonged storms. Intense storms during the spring of 1978 produced considerable flooding along East Fork Armells Creek, most notably adjacent to the north edge of the research area where an abandoned road fill obstructed the channel. During the late summer and autumn, no flow was observed, but ponded water in low spots in the channel suggested significant underflow in alluvial materials occurred throughout much of the year. The East Fork Armells Creek is classified as an intermittent stream.

Although several ephemeral drainages cross the site, surface flow was observed in these channels only during snowmelt. Mining operations temporarily obstructed the channels of two ephemeral drainages and produced short-lived impoundments which locally altered the ground-water recharge pattern in the area (Dollhopf et al., 1979).



*Includes Only Precipitation From Jan. 11,- Jan. 31, 1977

♦Colstrip NOAA Data
July 1979 & Later Data Collected by Western Energy Company

Figure 3 . Monthly precipitation recorded at the Western Energy Company Mine Area B, 1977 to 1980.

Two settling ponds were located in or adjacent to the research area. One pond, located at about N 49,000 - E 55,200 rarely contained water and had no observable effect on the hydrology of the research area. A second pond, at approximately N 48,000 - E 57,300 had considerable influence on ground-water recharge in the area. The pond received water pumped from the Mine Area B pit and a portion of the Mine Area E pit. Surface drainage from one ephemeral channel that was restored across spoil materials also drained into this pond but apparently contributed only a small percentage of the total inflow. Water from the pond was used for dust abatement on mine roads and no surface outflow was observed. Nearby well level fluctuations suggested a considerable amount of water was lost from the pond by subsurface seepage (Ground-Water section, pages 11 to 18).

The Area B mine pit has blocked most surface runoff from reaching spoil areas and has increased local infiltration by creating small impoundments in ephemeral channels. Recharge to spoil material will likely accelerate following highwall reduction and recontouring which will involve reconstruction of surface drainages across the mined area.

GROUND-WATER HYDROLOGY

The ground-water monitoring program was initiated during April, 1977 to obtain data for design and evaluation of the selective overburden placement demonstrations. Twenty-six wells were installed, each monitoring a specific zone: the Rosebud overburden, the Rosebud Coal, the McKay Coal and spoil materials. A number of piezometers were also installed and water level records were initiated for three wells previously installed by the Montana Bureau of Mines and Geology (Van Voast et al., 1977). The destruction of thirteen wells by mining and construction activities necessitated installation of nine additional wells west of the mine area during October 1978. To monitor the developing aquifer in spoil materials, a total of sixteen piezometers and two wells were installed in the mined area. Appendix A lists water level data and locations for both existing and destroyed wells. The following paragraphs describe the ground-water systems for the lithologic units above the base of the McKay Coal.

Rosebud Overburden

The diverse stratigraphy in the Rosebud overburden has apparently created varied ground-water conditions within this unit. Geologic materials ranged from massive cross-bedded sandstone to moderately thick shale (Geology section).

Generally, recharge to overburden materials appeared to be from vertical percolation of precipitation and surface runoff. Response of wells to local drainage obstructions indicated percolation was moderately rapid (Dollhopf et al., 1979). Overburden ground-water flow patterns roughly followed the topographic surface, with flow from higher plateau areas southwest of the mine towards the East Fork Armells Creek (Figure 4). The mine pit created a discharge area and locally, flow lines were diverted into the pit.

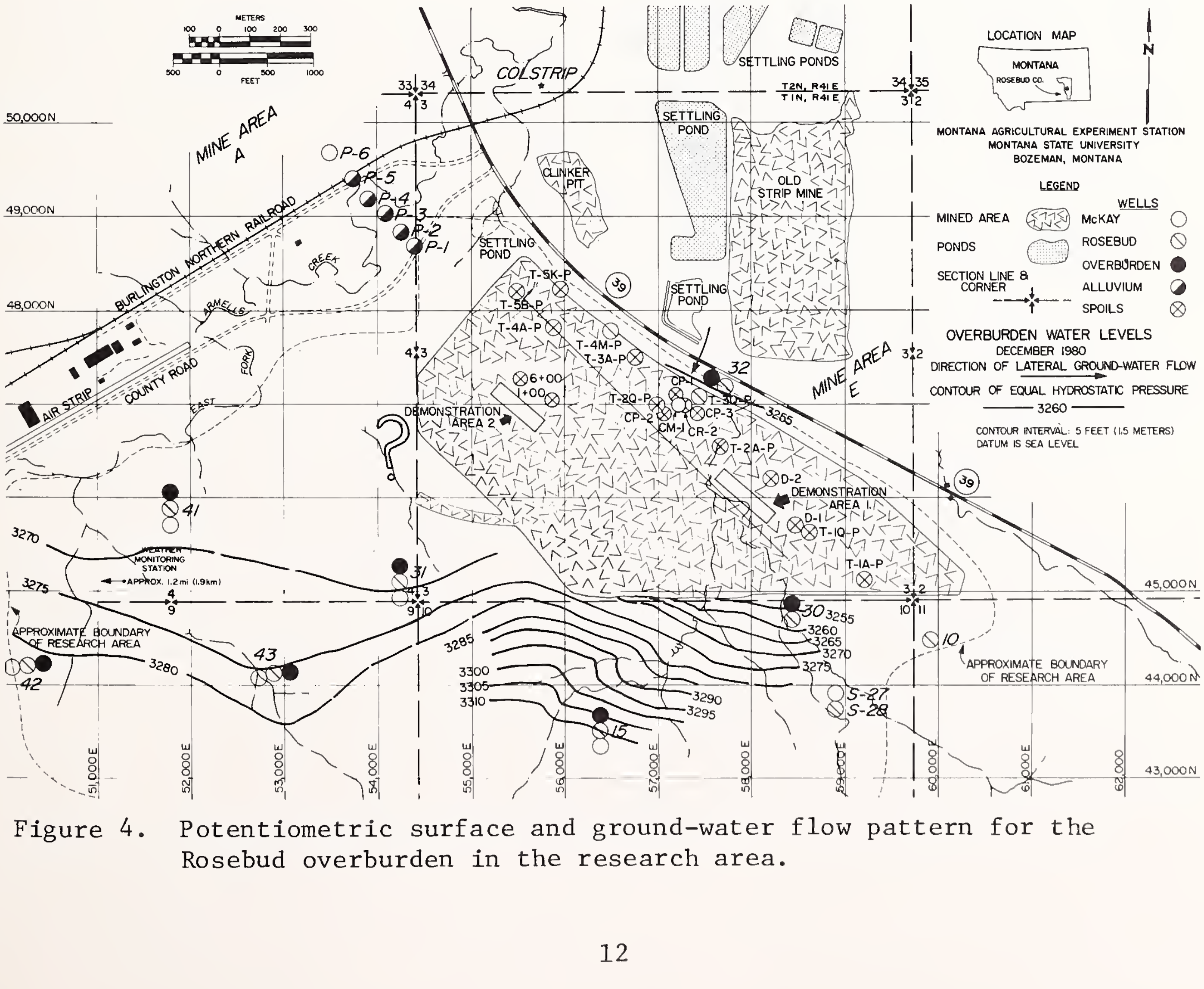


Figure 4. Potentiometric surface and ground-water flow pattern for the Rosebud overburden in the research area.

In areas adjacent to the East Fork Armells Creek, local recharge from the creek to overburden materials likely occurred during periods of high surface flows. This situation may have also occurred in areas where the mine pit cut off the natural down gradient flow or the presence of the pit may have locally reversed the natural gradient.

Well yields from 10 cm diameter wells monitoring zones of shale (15-0) and sandstone (31-0, 32-0) were about 2 L/min and 15 L/min respectively. A notable exception was well 43-0, in which the screened interval, with the exception of 15 cm of clinker, was entirely shale. This well yielded in excess of 35 L/min, possibly the result of fractures associated with minor faulting or from the clinker. Perched water tables likely exist above shale layers in the thick overburden headwater areas.

Wells 15-0 and 32-0 were equipped with continuous water level recorders. Records from these wells indicated little or no response to barometric pressure changes, suggesting unconfined or water table conditions existed for this unit in the research area. Sieve analyses indicated sandstone grain sizes of less than 250 µm or medium to very fine sand (see pages 34 to 36). Data from Johnson (1975) was used to estimate hydraulic conductivities which ranged from 0.9 m/day to 4.6 m/day (Table 1). In areas where shale dominated, this unit should be called an aquiclude or confining bed.

Table 1. Hydraulic parameters for aquifers in the research area.

SOURCE	METHOD	WELL	ZONE MONITORED	HYDRAULIC CONDUCTIVITY (K)m/DAY	TRANSMISSIVITY (T)m^2/DAY	STORAGE COEFFICIENT (S)
Dollhopf(a)	(1)	34-R	ROSEBUD COAL	.9-1.3	6.5-9.3	.000015
Dollhopf(a)	(2)	34-R	ROSEBUD COAL	.4	2.8	-
Van Voast(b)	(1)	S-28	ROSEBUD COAL	21	158	.00001
Dollhopf(a)	(3)	-	OVERBURDEN	.9-4.6	1.3-35	-
Van Voast(b)	(5)	S-27	MCKAY COAL	.58	1.39	-
Dollhopf(c)	(4)	D-2	SPOIL MATERIAL	-	-	.000014
This Report	(6)	CR-2	SPOIL MATERIAL	.01	.05	-

(a) Dollhopf et al., 1977

(b) Van Voast et al., 1977

(c) Dollhopf et al., 1978

(1) CONSTANT DISCHARGE PUMPING (Theis, 1935; Jacob, 1940; Johnson, 1975)

(2) RECOVERY DATA (Johnson, 1975)

(3) GRAIN SIZE-HYDRAULIC CONDUCTIVITY RELATIONSHIP (Lohman, 1972)

(4) BAROMETRIC EFFICIENCY (Jacob, 1940)

(5) BAILER RECOVERY (Skibitzke, 1958)

(6) INSTANTANEOUS RECHARGE (Cooper et al., 1967)

Mining has had a limited influence on overburden water levels. Pronounced drawdown was observed only at short distances from the mine pit, likely due to the poor aquifer characteristics, that is, discontinuous beds and low hydraulic conductivity values associated with the overburden.

Rosebud Coal

The Rosebud Coal is an aquifer over much of the Colstrip vicinity. Regional recharge patterns appear to be similar to overburden materials, with most recharge occurring in headwater areas (Van Voast et al., 1977). Shale layers in overlying overburden probably limit vertical recharge in some areas but, where channel sandstones directly overlie the coal, vertical recharge may be considerable. The regional flow pattern appears to be generally northeast through the research site (Van Voast (et al., 1977), but may be controlled locally by the stratigraphy and geologic structure (Figure 5).

The Rosebud Coal aquifer exhibits artesian conditions over much of the research area. Continuous water level recorders on wells 15-R and 32-R both indicated response to barometric pressure changes, suggesting confined conditions exist at these sites. Hydraulic parameters determined by pumping tests are indicated in Table 1. This aquifer is generally superior to the overburden in all hydraulic characteristics. This is due to its more homogenous

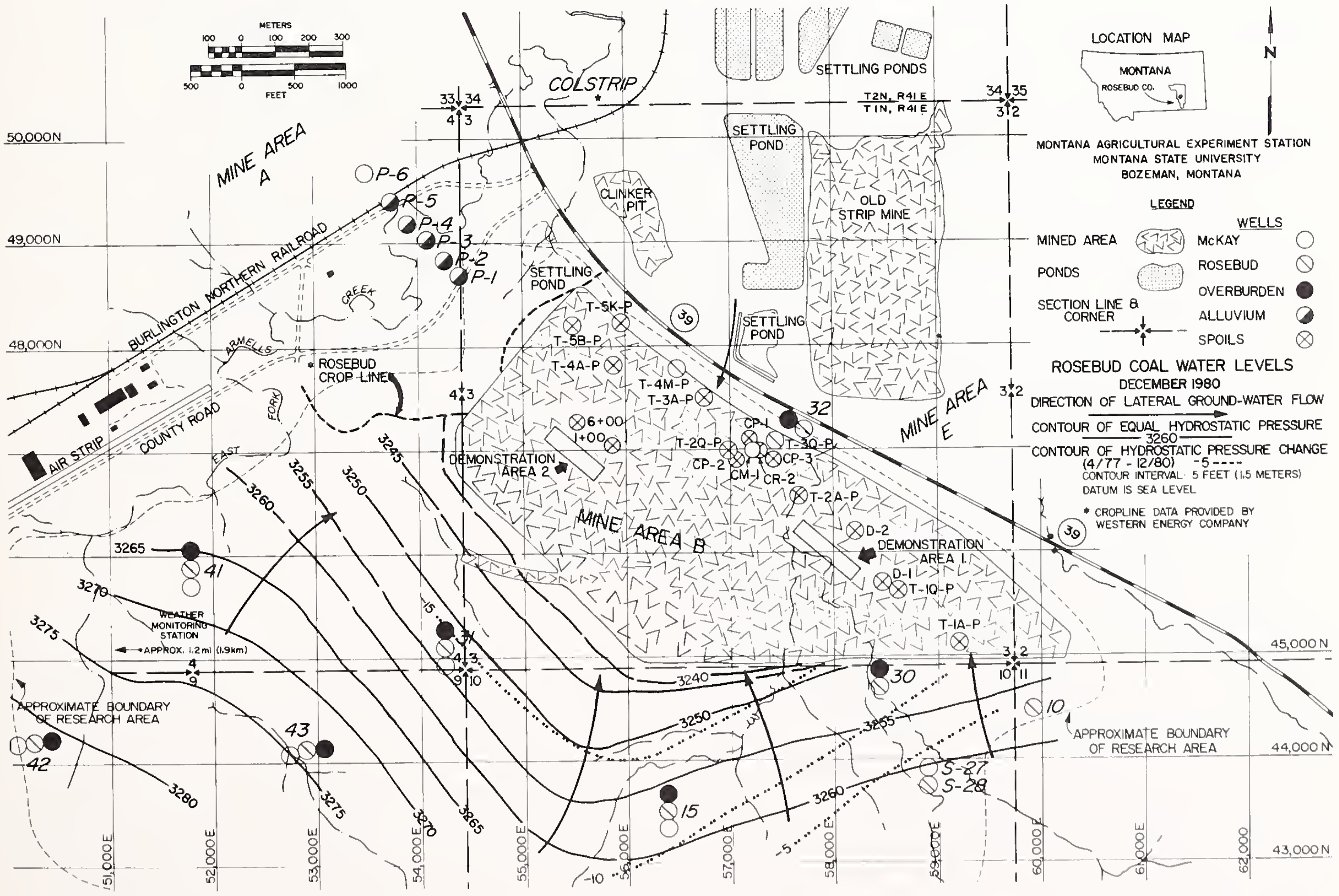


Figure 5. Potentiometric surface and ground-water flow pattern for the Rosebud Coal aquifer in the research area.

nature, that is, its more uniform thickness, composition, and lateral continuity. The hydraulic conductivity of the coal at any locality is dependent on the degree of fracturing (which is dependent on the jointing or the degree of folding and faulting in the area). A large degree of anisotropy may exist because of lineation of fractures or jointing and may explain some anomalous well yields.

Rosebud Coal water levels were influenced by mining to a greater degree than were water levels in any other aquifer. This was due in part to the greater hydraulic conductivity and transmissivity of the Rosebud Coal (Table 1), the steeper ground-water gradient established in the remaining coal by mining of the coal seam and, the more continuous nature of the coal.

The amount of drawdown produced by mining was most notable southeast of the mine pit (Figure 5). Pronounced decreases in water levels were observed at 457 m (1500 ft) from the mine pit in well 15-R. Some decreases in water levels were observed in wells 41-R and 43-R (Figure 5) during 1980 which may have been the result of prevailing drouth conditions rather than the result of mining activities. The position of the wells relative to East Fork Armells Creek may have had much bearing on the apparent drawdowns observed.

McKay Coal

Like the Rosebud Coal, the McKay Coal is laterally continuous over a wide area in the Colstrip vicinity. Although the seam is not mined in or adjacent to the research area, it is mined at the Peabody Coal Company's Big Sky Mine 4.8 km (3 mi) to the south.

Limited data suggests regional flow in the McKay aquifer is generally northeast, likely towards McKay outcrop areas north and east of Colstrip. In the immediate vicinity of the research area, the flow pattern is complicated by an apparent mounding of ground water beneath settling ponds adjacent to the northeast boundary of Mine Area B. Local recharge from the alluvium in the East Fork Armells Creek valley has also modified the regional flow pattern. Although the McKay Coal is not exposed at outcrops in or adjacent to the research area, drilling data indicated the bed subcrops in alluvium adjacent the north boundary of the research area (Dollhopf, et al., 1977b). The hydrographs of wells P-06, 41-M, and 42-M exhibit a response to spring runoff flows in the creek and usually peak during the March through May period. It is likely that some of the local recharge occurs from this source. The combined effects of local ground water mounding, the regional flow pattern and local recharge has created a southeast flow direction in the McKay Coal aquifer in the research area (Figure 6).

Records from a continous water level recorder on well 15-M indicated a water level response to barometric pressure changes which suggested the aquifer was confined at that location. This data was supported by the artesian conditions found in all wells monitoring the McKay Coal. The transmissivity value found for the McKay Coal was low. Van Voast et al (1977) found values of .58 m/day and 1.39 m^2/day for the hydraulic conductivity and transmissivity, respectively (Table 1). Well yields were generally less than 3.8 L (1 gal) per minute for most 10 cm wells. The transmissivity value was lower than respective values found for the Rosebud Coal due largely because the McKay Coal was less thick than the Rosebud Coal (2-3 m vs. 5.5-7.9 m).

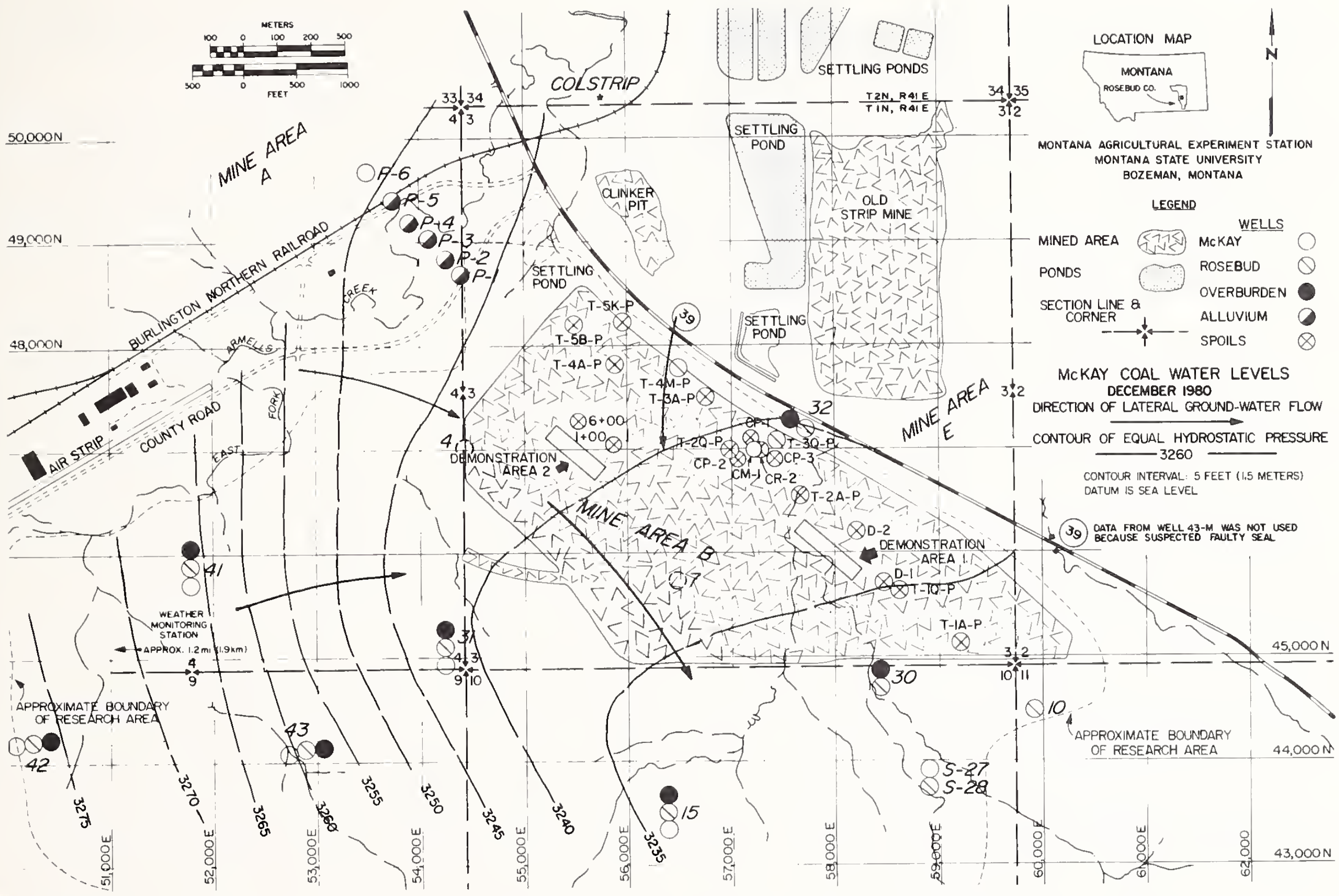


Figure 6. Potentiometric surface and ground-water flow pattern for the McKay Coal aquifer in the research area.

Mining has had little discernable effect on most wells monitoring the McKay Coal. However, a general decline in well 15-M and a very erratic hydrograph from well 31-M suggest some disturbance of this aquifer has occurred (Appendix A). It is possible that blasting in the area has increased leakage from this aquifer. It should be noted that both of these wells lie near the western edge of the mine pit in the direction of mine expansion (Figure 6).

Spoil Materials

A potentially important aquifer may be created by saturation of basal spoil materials in mined areas. Grazing is generally the projected post-mining land use and will require stockwater sources, which, due to insufficient surface supplies, will likely be derived from ground water. If mine spoil hydraulic characteritics are adequate, acceptable water may be derived from this source providing minimal water quality standards are met.

Data from wells and piezometers indicate most recharge to spoil materials occurs through the buried highwall on the northeast boundary of Mine Area B (Figure 1). Ground-water gradients near the buried highwall are somewhat greater than normal as the result of a ground-water mound formed beneath a settling pond in this vicinity (Dollhopf et al., 1978b, 1979). Drouth conditions desiccated the settling pond late in 1979 and hence, has decreased the effects of the mound. With completion of mining in Areas B and E, use of the settling pond will likely be discontinued. This may significantly decrease the recharge gradient from this direction in the future.

Recharge may also occur from the north. The East Fork Armells Creek, adjacent to the northwest end of Mine Area B, has eroded portions of the Rosebud and McKay Coal beds which both subcrop in alluvial materials (Dollhopf et al., 1977b). Ground-water levels in the alluvium (Van Voast et al., 1977), were significantly higher than the base of the spoil materials.

The ground-water gradient from the creek to the pit base, about .014, has likely induced flow from the creek and alluvium via the remaining coal and sandy overburden into the spoil materials.

Following mining and reclamation, ground-water levels in the spoil materials may approach the premining Rosebud Coal water levels. The reestablished ground-water flow will likely be down the topographic gradient, towards the East Fork Armells Creek.

Previous work (Van Voast et al., 1977) in the Colstrip area indicated hydraulic parameters for spoil materials did not differ greatly from those of the unmined overburden or coal. During August, 1979, development work was completed on well CR-2. Surging was used in combination with water jetting and over pumping for development. Although an increase in productivity was evident, yields remained too small (<3 L/min) to conduct a pump test. An eleven hour "slug" test was completed on the well in September 1979. Data from this test indicated spoil materials at this site may conduct water at a much slower rate than did the unmined coal (Table 1). It is probable that diverse conditions exist at the former mine pit base, that is, remaining unmined coal "fenders", rock rubble, and compacted areas. Therefore, the results of one test should not be considered typical of all spoil materials.

During this study, a total of five spoil monitoring wells and piezometers were equpipped with continuous water level recorders. Without exception, all exhibited a strong response to barometric pressure changes, suggesting widespread confined conditions exist for the reestablished spoil aquifer. Similar findings were reported for other mine areas in this vicinity (Van Voast et al., 1977). A storage coefficient of about 1.4×10^{-5} was determined from barometric efficiency data (Table 1). Spoil material compaction during the recontouring process probably produced the observed confinement.

Increases in spoil water levels within the mined area suggest the hydraulic conductivity of the bulk of these materials is probably in excess of that determined by testing well CR-2. It is probable that the apparent confined conditions will be present over most, if not all, the mined area. Eventually, following mining and the final highwall back filling, the ground-water gradient will probably revert to near premining conditions, with most recharge originating from the southwest and discharging to the north and northeast. If the hydraulic characteristics of the spoil materials differ considerably from that of the overburden materials, some adjustment in the piezometric surface should be expected, but it is unlikely that any significant change in the overall flow pattern will occur.

Summary

Several general conclusions can be drawn from the available data. These include: 1) Mining has had the most profound effect on the Rosebud Coal water levels. The response of this aquifer to mining is both more widespread and more rapid than the overlying overburden materials; 2) to date, there has been no recovery of Rosebud water levels adjacent to some reclaimed areas; 3) reestablishment of an aquifer in spoil materials is occurring at a uniform rate and ground-water levels may eventually approach the premine Rosebud Coal water levels; 4) the spoil aquifer exhibits apparent widespread confined conditions; and 5) the hydraulic conductivity of spoil materials may be locally much less than that of premine overburden and coal.

SURFACE WATER QUALITY

Surface water sources in semiarid Colstrip vicinity are limited by supply and quality. Aside from a few perennial water courses, most of the surface water sources are small, ephemeral tributaries and ponds. These water sources may exhibit substantial variations in water quality parameters. The chemistry of these water sources may be influenced by factors such as precipitation, surface runoff, ground-water discharges, freezing and thawing conditions, evaporation and regional geologic and soil characteristics of the watershed. Strip mining activities may influence the chemical composition of surface waters through mixing and redistribution of overburden materials, drainage channel alterations and pit dewatering activities. The East Fork Armells Creek is an intermittent tributary and is the principal surface water course in the research area. Most water samples of East Fork Armells Creek were collected during the spring and early summer when smowmelt and rain storms produced discharge events. The sampling location monitored on East Fork Armells Creek was approximately 2300 feet (channel distance) west of the point where State Highway 39 intersects the creek (about N 49350, E 53900, Figure 1). The location corresponds closely with sample location IN41E03 BBBB in Montana Bureau of Mines and Geology Bulletin 102 (Van Voast, et al., 1977). An impoundment utilized for mine waste water disposal was also sampled beginning in July, 1978. The impoundment was situated approximately 245 meters (804 ft) N of wells 32-0 and 32-R. Fluctuations of ground-water levels in wells 32-0 and 32-R strongly suggested the impoundment was recharging shallow aquifers in the area and thus, may have altered the ground-water quality in the area. Studies by Van Voast and Hedges (1976) on the reponse of shallow ground-water systems to surface hydrologic conditions at Colstrip indicated similar results.

Methods

Surface water samples were dip-sampled from East Fork Armells Creek on a quarterly basis from 1977 through June, 1978. During the July, 1978 through August, 1979 period, the impoundment and East Fork Armells Creek were sampled on a monthly schedule. Drought conditions limited surface water sampling during 1979 and 1980. Following sample collection, each 3785 ml (1 gal) sample was divided into four 500 ml (1 pt) bottles, filtered, acidified and refrigerated. The temperature, pH and EC (electrical conductivity) were determined in the field at the time of collection.

Following sample collection and preservation treatments, the water samples were placed in coolers with crushed ice and transported to the laboratory at Montana State University for chemical analyses.

A summary of the preservation treatments, corresponding laboratory analyses performed and laboratory procedures utilized on the water samples are referenced in Tables 2 and 3. The quality of surface and ground-water samples collected from the research area were evaluated by comparison with the water quality criterion established by the U.S. Public Health Service (USPHS) for public water supplies (Table 4). All analyses which exhibited a charge mass balance in excess of ±15 were eliminated from the data base. This procedure was performed to eliminate inaccurate analyses.

Table 2. Water sample treatment and corresponding laboratory analysis performed.

Preservation Treatments	Analyses Performed
Unfiltered, No Preservative	pH, Temperature, Conductivity, Bicarbonate, Carbonate, Fluoride, Total Phosphorous
Filtered, No Preservative	Boron, Chloride, Sulfate
Filtered, HNO_3	Calcium, Magnesium, Sodium, Iron, Manganese, Copper, Zinc, Lead, Cadmium, Nickel, Selenium
Unfiltered, H_2SO_4	Nitrate

Table 3. Laboratory procedures for water analyses (USEPA, 1974).

Element	Procedure
Pb, Cd, Cu, Fe, Zn, Mg,	Atomic Absorption Spectroscopy
Mn, Ca, Na, Ni, Mo, K	Atomic Absorption
Selenium	Atomic Absorption Spectroscopy H_2Se and AsH_3
pH	Electrode
Conductivity	Conductivity Bridge-Meter
HCO_3, CO_3	Titration
Sulfate	Turbidimetric
Total Phosphorous	Persulfate digestion - colorimetric
Boron	Curcumin Method
Nitrate-N	Cd reduction
Hardness (calculated)	Σ[Ca, Mg, Fe, Mn in mg/l)](constant)
Fluoride	Specific Ion Electrode
Chloride	Titration - Hg(NO_3)2
TDS (calculated)	Total cations + total anions + 0.4917 (HCO_3) - HCO_3 + F, K, Ni, V, Mo, Se and As (all in mg/l)
SAR (calculated)	$Na/[(Ca + Mg)/2]^{\frac{1}{2}}$, meq/l
Alkalinity (calculated)	(HCO_3, mg/l) 0.8202 + (CO_3, mg/l)3.31
Charge Mass Balance	Σ[Ca, Mg, Na] = Σ[SO_4, HCO_3, Cl] (all in meq/l)

Water Classification (major ionic constituents)

Following chemical analysis of the water samples, a general interpretation of baisc water type was determined through use of trilinear diagrams (Piper, 1944). Water types were classified based upon their overall ionic balance of cation and anion concentration percentages.

Table 4. Water quality criteria* (mg/l unless specified).

Constituent	Public Waters	Irrigation	Aquatic Life	Livestock
Boron	0.01	2.0	-	5.0
Cadmium	0.01	0.01	0.03	0.05
Chloride	250	100	-	1500
Copper	1.0	0.2	0.02	0.5
Iron	0.30	5.0	1.0	-
Lead	0.05	5.0	0.03	0.1
Manganese	0.05	0.5	1.0	10.0
Nitrate	10.0	-	-	100
pH	6.0-8.5		6.5-9.0	
Selenium	0.01	0.02	-	0.05
Sulfate	250	500(permissible)		500
Dissolved solids, total	500	500-1000	2000	3000
Zinc	5.0	2.0	-	25
Fluoride	-	1.0	1.5	2.0

* Criteria from: U.S. Dept. of Health, Educ. and Welfare (1962), U.S. EPA (1976), National Acad. of Sci. and Natl. Acad. of Engr. (1972), and McKee and Wolf (1963).

The large number of samples analyzed over a four year time span made it necessary to plot mean values for each diagnostic ion. Then a single overall water type was determined and plotted for each sampling site.

The mean characteristic water type, magnesium-calcium-sulfate (Mg-Ca-SO_4) was exhibited by both East Fork Armells Creek and the surface impoundment (Figure 7). The classifications were expressed by percentage of total cation + anion milliequivalents per liter (meq/l). Similar findings were reported by A.W. Hounslow and others (1978) for surface water samples inventoried at the Western Energy Mine near Colstrip. The dominant cation for East Fork Armells Creek and the surface impoundment was magnesium, based upon percentage of total cation meq/l. Calcium and sodium were relatively interchangeable as secondary cationic constituents. The predominant anion represented by both surface water sources was sulfate (SO_4), also expressed as a percentage of the total anion meq/l. The water analyses showed that the constituents Ca, Mg, Na, SO_4 and HCO_3 represented at least 98% of the total ions in solution for the surface and ground waters sampled.

Conductivity and TDS

Specific conductance is generally a good indicator of the degree of mineralization inherent in a water sample. The mean conductance value for East Fork Armells Creek was 1750 micromhos per centimeter (μmhos/cm), indicating a potentially high salinity hazard if used as an irrigation source. The average value represented by the settling pond (impoundment) was somewhat less, at 1106 μmhos/cm, but still presented a moderately high salinity hazard. Both surface waters exhibited substantial ranges for conductivity. East Fork Armells Creek ranged from 95-2860 μmhos/cm while the impoundment ranged from

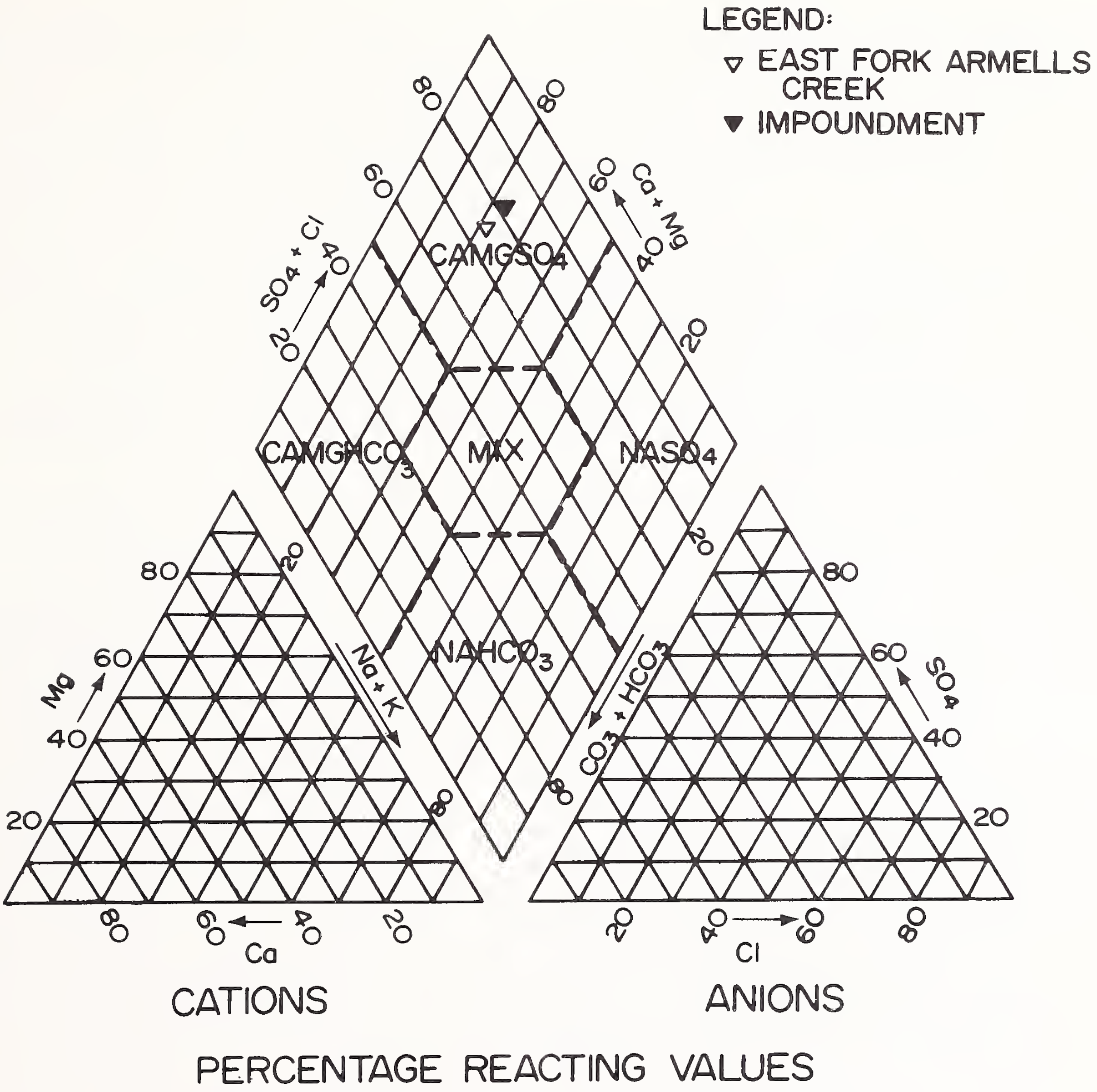


Figure 7. Trilinear diagram exhibiting the mean relative chemical compositions of surface water sampled from East Fork Armells Creek and the impoundment, Western Energy mine Area B, Colstrip, Montana.

160-1920 µmhos/cm (Table 5). The variations in concentration of the major chemical parameters over time are denoted in Appendix B.

Mean total dissolved solids (TDS) for East Fork Armells Creek and the impoundment were 1493 mg/l and 882 mg/l respectively. The corresponding range of TDS values were 89-2724 mg/l and 163-1612 mg/l. Through examination of the chemical data for both surface waters, a gradual increase in mineralization each year from spring through the summer months was evident. It was more obvious in the surface impoundment than in East Fork Armells Creek that a definite pattern of salt enrichment occurred. The increased concentration of salt in the impoundment was attributed to the intensified evaporation during summer months. As the water level dropped in the impoundment, the water temperature warmed, accounting for the rise in pH, elevated CO_3 disso-

Table 5. Ranges and means for all water sample sources in the MAES research area, Western Energy Company Mine Area B, near Colstrip, Montana.

Sample Location		Chemical Parameters (mg/1)																				
		EC (field) µmhos/cm	pH (field)	TDS	Alkalinity (CaCO$_3$)	Hardness (CaCO$_3$)	Ca	Mg	Na	HCO$_3$	SO$_4$	Fe	Cu	Zn	Mn	Pb	Cd	F	Cl	NO$_3$-N	Total P	SAR
Surface Water																						
Armell's Ck	(R)	95-2860	7.6-8.4	89-2724	20-615	48-1841	10-228	6-349	3-248	24-750	30-1716	<.010-.270	<.010-.495	<.005-.036	.010-5.70	<.010-.030	<.005-.015	<.01-.30	<.01-29.6	<.01-.35	<.01-.05	.20-.307
	($\bar{X}$)	1750	7.9	1493	298	901	118	147	111	354	924	.072	.092	.017	.629	.011	.005	.202	10.53	.089	.018	1.44
Settling Pond	(R)	160-1920	6.6-9.1	163-1612	58-238	114-801	26-109	12-152	9-147	70-265	79-1148	<.010-.50	<.010-.042	.010-.022	.005-.036	<.010	.005-.007	.12-.21	0.9-12.7	<.01-.53	.004-.02	.37-2.26
	($\bar{X}$)	1160	8.1	882	145	532	73	85	74	159	560	.025	.020	.014	.012	<.010	.005	.184	5.64	.156	.012	1.32
Rosebud Wells																						
4-R	(R)	1310-1560	7.0-7.6	1198-1306	455-515	792-923	134-166	105-123	79-132	506-580	543-656	.016-.070	.011-.032	.180-.598	.180-.542	<.010-.015	<.005	.20-.30	5.2-9.4	<.01-.06	0-.09	1.20-2.00
	($\bar{X}$)	1437	7.3	1249	475	852	152	115	96	562	595	.040	.024	.293	.298	.011	<.005	.244	7.8	.028	.031	1.43
5-R	(R)	2610-3100	7.0-7.1	2563-2988	685-738	1660-2107	195-225	285-375	160-165	836-900	1396-1693	.074-.110	.019-.030	.029-.438	.176-.547	<.010-.012	<.005	.17-.18	16.90-18.30	.03-.14	0-.01	1.6-1.7
	($\bar{X}$)	2855	7.0	2766	712	1884	210	330	162	868	1544	.092	.025	.234	.478	.011	<.005	.175	17.60	.085	.005	1.65
7-R	(R)	1330	7.6	1134	285	680	122	91	107	348	636	.030	.012	.026	.156	.015	.012	.20	6.60	.03	0	1.80
	($\bar{X}$)	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
10-R	(R)	1550-2900	6.9-7.3	1434-2552	690-943	625-1607	95-231	92-259	209-323	868-1150	593-1257	<.010-.121	.009-.058	.038-.551	.134-.608	<.010	<.005	.06-.20	3.3-11.3	<.01-.33	.001-.08	2.80-4.39
	($\bar{X}$)	2109	7.1	1883	803	1027	160	152	254	976	820	.051	.029	.211	.300	<.010	<.005	.12	6.8	.15	.02	3.54
15-R	(R)	1340-2000	7.1-9.9	1246-1487	433-625	798-1048	120-183	110-168	78-201	528-763	501-744	.010-.069	.010-.080	.033-.240	.079-.321	<.010	<.005	.01-.17	2.9-13.2	<.01-.30	.001-.06	1.12-2.72
	($\bar{X}$)	1634	7.4	1406	542	925	149	134	113	661	668	.040	.028	.117	.157	<.010	<.005	.091	10.48	.065	.018	1.62
28-R	(R)	1070	8.9	857	95	499	54	88	77	104	579	.020	.010	.005	.028	.010	.011	.24	5.20	.04	0	1.50
	($\bar{X}$)	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
29-R	(R)	805	7.4	636	398	456	62	74	40	485	220	.022	.010	.017	.148	.015	<.005	.20	1.40	.03	0	.80
	($\bar{X}$)	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
30-R	(R)	660-1400	7.1-8.3	434-840	306-650	388-663	37-110	70-130	18-40	348-793	122-345	<.010-.076	<.010-.063	.016-.448	<.010-.170	<.010-.010	<.005-.005	.07-.33	.50-.3.3	<.01-.53	0-.08	.34-.72
	($\bar{X}$)	880	7.6	668	375	548	56	99	27	446	259	.031	.026	.084	.074	.010	.005	.231	2.04	.156	.026	.46
31-R	(R)	1600-2700	6.8-7.4	1763-2300	485-663	761-1578	105-231	118-233	108-342	592-808	810-1364	.012-.120	.012-.080	.028-.188	.132-.331	<.010-.020	<.005-.014	.04-.34	3.6-7.5	<.01-.50	0-.07	1.30-5.40
	($\bar{X}$)	2210	7.1	2064	449	1273	175	197	172	710	1154	.046	.030	.081	.241	.011	.006	.191	5.56	.104	.023	2.15
32-R	(R)	870-1550	7.0-7.8	830-1095	308-499	451-757	70-112	65-132	59-112	375-608	280-562	<.010-.118	<.009-.090	.099-.218	.064-.302	<.010-.015	<.005-.008	.04-.30	2.4-9.4	<.01-.59	0-.09	.9-1.91
	($\bar{X}$)	1146	7.3	928	452	594	98	88	85	521	374	.038	.026	.074	.158	.010	.005	.182	4.19	.113	.022	1.50
33-R	(R)	1620-2000	7.1-8.4	1245-1730	280-513	724-1114	102-180	114-169	101-161	342-625	648-904	<.010-.100	<.009-.020	.055-.238	.046-.200	<.010	<.005	.03-.20	7.1-9.9	.01-.53	0-.02	1.5-2.11
	($\bar{X}$)	1781	7.5	1485	425	955	142	146	126	510	788	.054	.013	.124	.105	<.010	<.005	.162	8.36	.225	.011	1.77
34-R	(R)	2110-2800	7.0-7.1	1936-2377	543-605	1103-1585	170-277	165-218	149-190	662-738	1113-1337	.075-.190	<.005-.026	.043-.163	.090-.196	<.010-.035	<.005-.008	.16-.20	11.8-14.1	.06-.21	0-.02	2.0-2.1
	($\bar{X}$)	2402	7.1	2236	585	1442	244	202	179	714	1238	.138	.012	.074	.156	.021	.006	.182	12.95	.122	.01	2.08
41-R	(R)	1030-1850	6.9-7.5	1048-1310	432-512	533-979	74-136	82-155	92-188	458-624	405-578	<.010-.258	<.010-.036	.080-.286	.046-.434	<.010-.010	<.005-.012	.04-.28	7.5-11.7	<.01-.15	.001-.07	1.50-2.91
	($\bar{X}$)	1353	7.3	1144	482	721	104	112	113	581	513	.078	.019	.159	.174	.010	.006	.193	9.08	.069	.024	1.83
42-R	(R)	700-1150	7.2-7.6	587-882	373-670	388-608	58-98	46-88	38-72	454-817	134-296	<.010-.128	<.008-.066	.013-.297	.106-.325	<.010-.018	<.005	.10-.24	1.4-4.4	<.01-.11	<.01-.09	.76-1.58
	($\bar{X}$)	867	7.4	669	460	463	78	65	56	560	185	.053	.025	.107	.219	.012	<.005	.170	3.07	.038	.035	1.15
43-R	(R)	1410-2420	6.9-7.7	1389-2326	441-713	817-1509	96-261	140-218	98-179	538-869	725-1328	.028-.221	.010-.048	.005-.500	.190-.380	<.010	<.005	.07-.19	3.3-7.1	<.01-.15	.01-.07	1.49-2.30
	($\bar{X}$)	1989	7.2	1840	636	1154	182	170	150	775	944	.114	.022	.146	.292	<.010	<.005	.148	5.54	.052	.026	1.93
McKay Wells																						
4-M	(R)	1460-1700	7.2-7.8	1205-1472	416-495	611-910	116-166	78-120	142-189	508-604	496-717	.020-.086	.007-.023	.019-.330	.095-.208	<.010-.020	<.005-.010	.18-.24	3.3-6.1	<.01-.09	0-.01	1.80-3.02
	($\bar{X}$)	1576	7.4	1322	464	757	142	98	172	556	624	.059	.017	.087	.134	.011	.006	.21	4.9	.059	.009	2.61
7-M	(R)	1720	7.7	1512	330	743	117	109	170	408	912	.080	.012	.060	.102	<.010	.008	.24	5.2	.005	0	2.70
	($\bar{X}$)	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
15-M	(R)	1680-3200	7.0-7.7	1756-2651	461-901	920-1398	154-271	130-211	127-330	532-1098	942-1515	.010-.092	<.010-.068	.068-.443	.082-.180	<.010	<.005	.02-.19	4.2-11.3	<.01-.25	<.001-.07	1.60-6.21
	($\bar{X}$)	2418	7.2	2292	549	1213	212	166	259	663	1307	.045	.030	.168	.134	<.010	<.005	.094	6.72	.079	.024	3.45
31-M	(R)	1900-2800	6.8-7.4	1807-2246	517-730	759-1571	136-204	96-271	121-315	473-889	915-1288	.015-.824	.012-.061	.040-.545	.042-.310	<.010	<.005	.17-.37	3.8-6.2	<.01-.26	.004-.08	1.89-5.09
	($\bar{X}$)	2272	7.2	2053	581	1066	174	153	254	689	1124	.194	.030	.215	.100	<.010	<.005	.302	4.76	.088	.023	3.70
41-M	(R)	880-1300	7.4-11.7	696-996	280-480	65-199	6-55	3-38	144-252	22-453	236-413	<.010-.122	<.008-.035	<.005-.080	<.010-.100	<.010	<.005	.24-.42	2.8-5.2	.01-.13	<.01-.11	5.12-16.80
	($\bar{X}$)	1015	9.2	763	361	127	22	15	224	259	291	.050	.020	.023	.029	<.010	<.005	.36	3.97	.052	.053	10.02
42-M	(R)	700-1200	7.2-7.9	657-789	448-515	162-367	45-68	27-48	137-187	546-628	121-208	<.010-.072	<.008-.038	.086-.275	.060-.164	<.010	<.005	.08-.20	2.4-3.8	<.01-.13	.012-.07	3.20-4.25
	($\bar{X}$)	928	7.5	717	477	278	57	37	151	582	176	.032	.020	.157	.109	<.010	<.005	.153	2.97	.047	.028	3.85
43-M	(R)	1440-2720	7.0-7.8	1394-2024	539-751	810-1415	106-225	131-207	129-215	519-915	729-972	.010-.284	<.008-.053	.050-.905	.037-.290	<.010	<.005	.07-.24	2.8-7.5	<.01-.18	<.01-.07	1.89-2.75
	($\bar{X}$)	1956	7.2	1760	701	1029	163	151	180	815	851	.086	.022	.346	.197	<.010	<.005	.16	5.68	.07	.022	2.45
CM-1	(R)	990-1750	6.9-7.5	927-1293	285-491	540-738	98-156	74-207	88-152	348-599	379-709	.010-1.084	<.010-.051	.014-1.098	.247-.684	<.010	<.005	.10-.23	3.3-7.5	<.01-.30	<.01-.19	1.48-2.48
	($\bar{X}$)	1331	7.4	1133	397	661	120	87	123	485	549	.311	.035	.177	.449	<.010	<.005	.18	5.64	.082	.036	2.09
Overburden Wells																						
7-0	(R)	1710	7.3	1607	548	1144	120	205	87	668	803	.060	.019	.039	.206	.015	<.005	.20	8.00	.01	.02	1.10
	($\bar{X}$)	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
15-0	(R)	1800-2700	6.8-9.9	1827-2388	520-600	1064-1575	162-278	90-230	148-359	634-732	960-1653	<.010-.322	.010-.060	.007-.592	.009-.520	<.010	<.005-.014	.03-.26	4.2-11.3	<.01-.47	<.01-.11	1.90-4.01
	($\bar{X}$)	2264	7.4	2148	541	1270	215	178	187	658	1261	.058	.033	.216	.194	<.010	.006	.126	8.3	.133	.020	2.28
28-0	(R)	1220	7.6	977	400	700	75	124	61	488	469	.058	.016	.025	.038	<.010	.009	.22	6.60	.12	.13	1.00
	($\bar{X}$)	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
29-0	(R)	525	7.8	345	265	322	45	51	11	323	77	.022	.019	.005	.040	<.010	.011	.22	.90	.01	.51	.30
	($\bar{X}$)	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
30-0	(R)	490-960	7.1-8.0	335-655	290-357	306-521	27-60	58-90	15-36	336-422	109-304	<.010-.078	<.009-.104	.019-.111	<.005-.117	<.010-.013	<.005	.16-.40	.9-2.4	<.01-.40	<.01-.02	.34-.74
	($\bar{X}$)	699	7.6	500	327	423	42	76	20	388	149	.030	.043	.054	.035	.010	<.005	.322	1.44	.155	.012	.44
31-0	(R)	1700-2600	6.8-7.5	1708-2185	613-820	1083-1614	95-228	166-278	108-245	747-1000	713-1202	<.300-1.184	<.009-.045	<.055-.275	.040-.175	<.010-.025	<.005-.012	.03-.22	3.8-8.7	<.01-.26	<.01-.05	1.26-2.77
	($\bar{X}$)	2140	7.1	1970	684	1320	165	221	134	834	1027	.661	.025	.076	.099	.011	.006	.161	6.39	.049	.018	1.62
32-0	(R)	1900-3570	7.0-8.0	1910-3098	285-425	1076-1787	94-243	180-297	152-281	348-519	1208-4051	<.010-.116	<.010-.080	.028-.377	.004-.288	<.010	<.005-.010	.05-.26	6.1-9.9	<.01-.74	<.01-.11	1.88-3.17
	($\bar{X}$)	2591	7.4	2496	339	1414	190	228	228	428	1612	.047	.030	.087	.137	<.010	.005	.195	8.00	.226	.022	2.65
33-0	(R)	965-1360	7.3-10.7	916-1040	265-390	694-876	78-105	111-150	53-112	299-433	431-535	<.010-.130	<.005-.040	.005-.197	<.005-.062	<.010	<.005	.05-.31	6.1-10.8	.12-.76	<.01-.03	.89-1.79
	($\bar{X}$)	1192	8.0	990	361	770	93	129	66	395	477	.053	.018	.097	.029	<.010	<.005	.211	8.06	.284	.016	1.08
41-0	(R)	1070-1860	7.1-8.5	1123-1430	190-693	547-1112	74-150	80-179	89-134	232-845	596-759	<.010-1.280	<.008-.045	<.005-.166	.032-.200	<.010	<.005	.11-.30	9.8-15.1	.02-.14	<.01-.05	1.31-1.93
	($\bar{X}$)	1455	7.7	1268	381	600	108	127	105	464	679	.151	.028	.043	.096	<.010	<.005	.201	12.7	.055	.017	1.63
42-0	(R)	720-950	7.3-7.6	558-663	353-421	329-584	51-143	49-75	27-44	430-513	137-217	<.010-.102	<.008-.036	.010-.215	.188-.406	<.010	<.005	.17-.43	2.4-4.7	<.01-.10	<.01-.05	.54-.94
	($\bar{X}$)	801	7.5	598	380	452	79	62	34	463	187	.042	.020	.069	.268	<.010	<.005	.329	3.71	.038	.019	.70
43-0	(R)	1200-2090	6.9-7.4	1136-2040	350-698	786-1265	92-170	127-204	63-137	427-851	619-1120	.015-.189	<.010-.066	.029-.091	.066-.275	<.010	<.005	.12-.31	6.1-10.4	<.01-.08	<.01-.06	.98-1.81
	($\bar{X}$)	1697	7.1	1590	512	1037	140	167	92	625	861	.074	.025	.048	.152	<.010	<.005	.20	7.62	.031	.021	1.36
Spoil Wells																						
CR-2	(R)	3210-4400	6.8-7.7	3196-4933	248-360	1910-2958	270-426	275-460	319-412	302-439	2050-3553	<.010-.126	.020-.075	.080-.835	.207-.710	<.010	<.005-.020	.09-.29	40.9-51.8	<.01-.30	<.01-.05	3.07-3.51
	($\bar{X}$)	3737	7.4	4398	302	2331	348	369	363	368	3067	.056	.044	.371	.407	<.010	.007	.204	47.59	.149	.022	3.26
D-2	(R)	1870-2010	7.2-7.4	1850-2025	432-453	1207-1407	173-202	188-219	151-164	522-552	1036-1164	.028-.090	.012-.024	.022-.026	.324-.378	<.010	<.005	.20	14.1-16.5	.007-.01	.01	1.75-2.06
	($\bar{X}$)	1920	7.3	1947	446	1294	185	202	156	553	1121	.049	.019	.024	.359	<.010	<.005	.20	15.53	.009	.01	1.90

lution, HCO_3 reduction, and probable precipitation of calcite ($CaCO_3$) with a corresponding enrichment of Mg, Na, Cl, K, and SO_4. A similar chemical enrichment trend was observed in the East Fork Armells Creek data (Appendix B). Once again an increase in mineralization was prevalent as the seasonal runoff volume decreased and evaporative processes concentrated the ionic constituents in solution. The repetitive instances of a relatively high degree of mineralization which were apparent in May 1977, April 1978, and April 1979 were enigmatic. During these periods of high soil moisture, more mineral salts may have been leached from the local surface and subsurface soil profile and carried into the stream channel as surface runoff and shallow seeps. It was also likely that concentrated salts left within the stream channel from the previous dry season may have been incorporated into solution with the initial spring melt, thus contributing to the elevated chemical concentrations evident at this time period.

pH, Alkalinity

The mean pH values for the two surface waters monitored were 7.9 (Armells Creek) and 8.1 (impoundment). The ranges in pH values were 7.6-8.4 and 6.6-9.1 respectively. The pH level is directly influenced by the solubility of numerous ionic constituents in an aqueous solution. A change in pH may reflect corresponding changes in the alkalinity and hardness levels of a water source.

Alkalinity is a measure of the buffering capacity of a water and is commonly determined from the ionic concentrations of HCO_3, CO_3, total P and hydroxides. Natural surface and ground-water sources are generally low in phosphate and hydroxide concentrations, consequently HCO_3 and CO_3 make up the bulk of an alkalinity value. Bicarbonate and CO_3 anions and some clays will commonly complex some toxic heavy metals and reduce their toxicity. Alkalinity is generally not considered a concern to public health unless values are consistently in excess of 400 mg/l as $CaCO_3$. Irrigation waters containing alkalinity values in excess of 600 mg/l may produce symptoms of chlorosis in sensitive plants when the iron is precipitated out of solution as a hydroxide and becomes unavailable to the plant. Other carbonate precipitates of Ca and Mg may also occur if the bicarbonate concentration is high. This condition, when prevalent, may result in a sodium-rich solution with a correspondingly higher SAR, which may result in soil or plant damage (U.S. EPA, 1976). The mean alkalinity value for East Fork Armells Creek was 298 mg/l with a corresponding range of 20-615 mg/l. Of 15 different sampling dates, 8 of the total exhibit alkalinity concentrations >400 mg/l. The alkalinity values should pose little threat to vegetation production although potential problems might arise if the water is used for domestic purposes. The mean alkalinity value for the impoundment was 145 mg/l and the range was 58-238 mg/l as $CaCO_3$, and thus, well below the suggested 400 mg/l limit.

Hardness + SAR

Calcium and Mg cations, with a lesser contribution attributable to Fe, Sr and Mn generally determine the hardness as $CaCO_3$. The majority of the hardness values for both surface water sources were very hard (>300 mg/l as $CaCO_3$). Mean values were 901 and 532 mg/l for the East Fork Armells Creek and the impoundment respectively.

Typical range values were 48-1841 mg/l for the East Fork Armells Creek and 114-801 mg/l for the impoundment (Table 5). Hardness is not usually a concern for irrigation water use, although the concentrations of the cations, Ca and Mg are important in determining the exchangeable sodium potential. Generally, hardness in water has not proven to be of serious concern to public health and consequently, the established standards are a result of economic principles (U.S. EPA, 1976).

Sodium adsorption ratio (SAR) values for both surface water sites were low. The average SAR value for East Fork Armells Creek was 1.44 with a range of 0.20-3.07. The SAR values for the impoundment ranged from 0.37-2.26 and averaged 1.32. The highest SAR values were observed during the drier, warmer months when evaporation concentrated the total dissolved solids (Appendix B). There should be little risk of sodic soil development if these surface waters are utilized as an irrigation source, based upon the SAR values recorded (U.S.D.A., 1969).

Minor + Trace Elements

Relatively few minor or trace element concentrations were of concern in the surface waters sampled. The elemental concentrations present in the water samples were compared to guidelines suggested by the USPHS for domestic water supplies. Manganese (Mn) was the only trace element to frequently exceed the recommended limits (0.05 mg/l) in waters sampled from East Fork Armells Creek. The Mn criterion was exceeded in approximately 69% of the samples analyzed. The concentrations ranged from 0.01-5.70 mg/l with a mean value of 0.63 mg/l. The source of the manganese may be attributable to the weathering and dissolution of manganite, pyrolusite or psilomelane. There were no instances where Mn exceeded the established maximum criteria in the waters from the impoundment.

Additonal chemical parameters exceeding the USPHS standards for domestic water from both surface water sources were SO_4 and TDS. Sulfate and TDS concentrations in water from East Fork Armells Creek and the impoundment exceeded the suggested limits of 250 and 500 mg/l, respectively, in 75% of the sample analyses. The exact elemental concentrations obtained from the respective runoff analyses for both sampling sites are presented in Appendix B.

GROUND-WATER QUALITY

It may be stated with some certainty, that the characteristic chemicals found in most ground water are principally derived from the water soluble materials comprising the local and/or regional geologic materials and subsurface stratigraphy through which the water must move. As the sedimentary rocks typical of the Tongue River Member of Fort Union Formation are crushed during strip-mining activities, fresh mineral surfaces are exposed to the percolation of ground waters which originate as infiltrating surface waters. As rock types, previously isolated below the water table or between aquifers, are emplaced in a subaerial environment due to spoiling practices, accelerated weathering reactions may occur. Inevitably, the resultant hydrogeochemical reactions will change the hydrochemical balance of the ground water to some degrees (Rahn, 1976).

Methods

Ground-water samples were collected from a series of observation wells completed within the overburden, spoils, Rosebud and McKay Coal aquifers at research site (Figure 1).

Samples were obtained from the wells with a 4-inch submersible pump. Initially wells were pumped on a quarterly basis (April 1977-April 1978). A monthly sampling schedule was instrumented during June 1978 to monitor potential changes in water chemistry more closely. Following August 1979, samples were again acquired on a quarterly basis. Water quality analyses were performed as described for surface water samples in the previous section. Once again, all analyses exceeding a charge mass balance of ±15 were eliminated from the data base.

Water Classification (major ionic constitutents)

Although there were frequent differences observed in the actual elemental concentration levels in water samples (mg/l) between wells, there were few wells which could not be classified as Mg-Ca-SO_4 water type based upon the percentage of total cation and anion milliequivalents per liter. Again, these were mean representative plots for each well.

Of the 11 overburden wells monitored, 3 wells (29-0, 30-0, and 42-0) exhibited a Mg-Ca-HCO_3 water type (Figure 8). Three of the 15 Rosebud Coal wells (29-R, 30-R, and 42-R) were also representative of the Mg-Ca-HCO_3 category (Figure 9). The dominance of HCO_3 in some samples may have been the result of sulfate reduction. In ground waters where water and organic material (i.e., coal seams) are in contact, sulfate reduction may occur by the following reaction, ($SO_4^{=}$ +$CH_4 \rightleftharpoons HS^{-}$+$HCO_3^{-}$+$H_2O$) which is catalyzed by anerobic bacteria (Hounslow, et al., 1978). Following the reduction of the

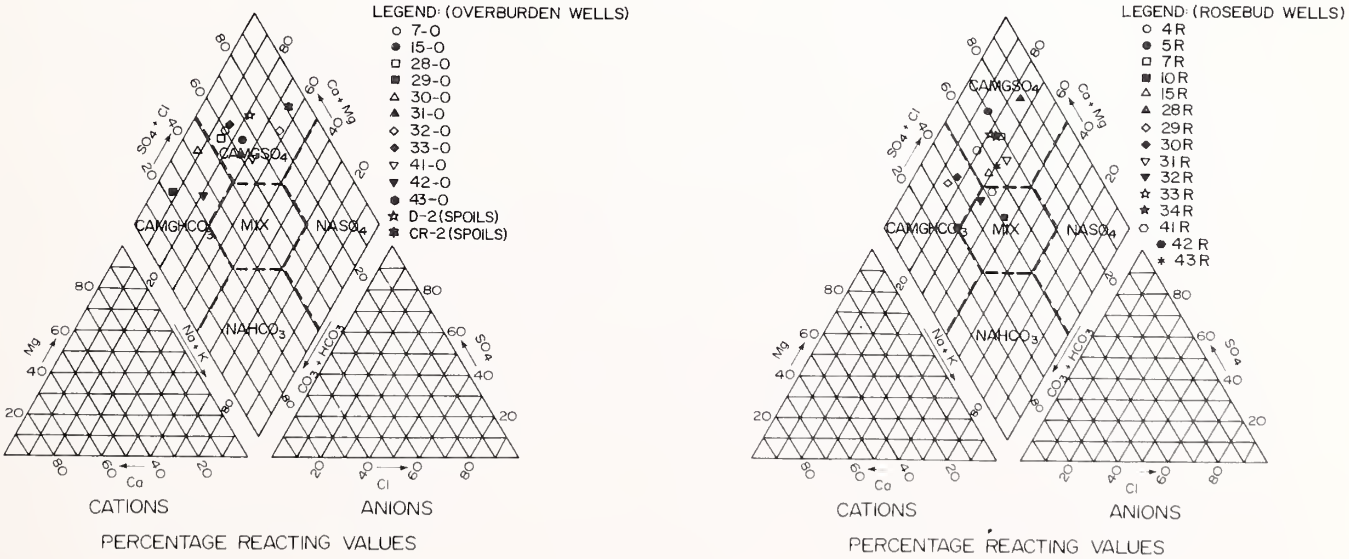


Figure 8. Trilinear diagram classification of mean values for overburden and Rosebud water samples, Western Energy Company Mine Area B, Colstrip, Montana.

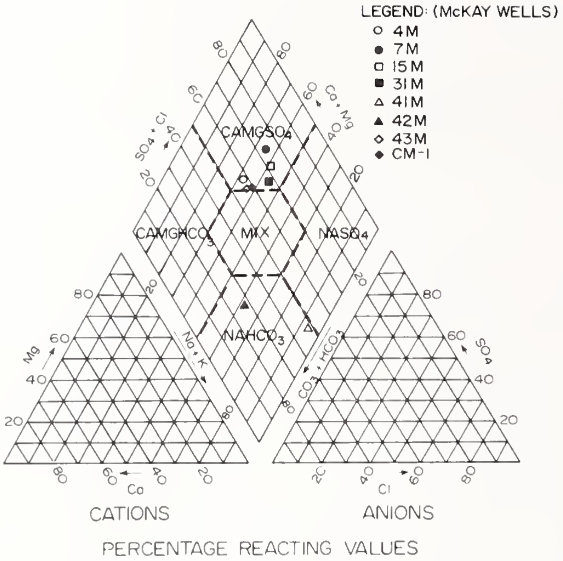


Figure 9. Trilinear diagram classification of mean values for McKay water samples, Western Energy Company Mine Area B, Colstrip, Montana.

sulfate ions and oxidation of organic carbon by sulfur-reducing bacteria, hydrogen sulfide gas (H_2S) and hydrogen sulfide ions (HS^-) are generated as by-products and dispersed into the aqueous solution. If any iron is present in the water under these reducing conditions, iron sulfide (pyrite, marcasite) may be precipitated, thus removing the iron and sulfide from the water. Reducing conditions are likely limited as most ground waters usually have at least some trace amounts of oxygen present which would readily precipitate any ferrous iron as amorphous ferric hydroxide ($Fe(OH)_3$). This is especially true in the pH range of 7-8, typical of the waters in arid regions (Davis and DeWiest, 1967). Several of the aforementioned wells exhibited the probable presence of sulfate reduction, indicated by the presence of H_2S gas.

Sulfate bacteria may also be active in the soil profile, and if present in areas where recharge waters have percolated, then sulfate concentration may be reduced in the corresponding aquifer to low levels. This may explain the HCO_3 dominance in the shallower overburden wells mentioned above.

Another atypical water type (Na-HCO_3) was exhibited by wells 41-M and 42-M (Figure 9). In the samples from well 42-M, the percentage of Na in meq/l was substantially higher than the Ca + Mg values. The HCO_3 concentration was compatible with the levels found in the Mg-Ca-HCO_3 water type. Again, the HCO_3 predominance may have been attributable to SO_4 reduction within the aquifer. One possible explanation for the reduced levels of Ca and Mg and elevated Na, may have been due to the adsorption of Ca and Mg by smectite clay with the subsequent release of Na cations into solution. Smectite was one of several clays identified by X-ray diffraction techniques from spoil and overburden samples collected within the mine area (Schafer et al., 1979; Dollhopf et al., 1978b and Hounslow et al., 1978).

Well 41-M was constructed in November 1978 and its water chemistry was quite enigmatic. Appendix B illustrates the analytical results of the water sampled from 41-M. Typically, Na and HCO_3 were the dominant cation and anion respectively. Carbonate concentrations were elevated as the result of the high pH values (9-11) which caused dissociation of HCO_3^- ions to CO_3 ions. A gradual decrease in the pH over time has had an effect on the overall ionic balance. Although the cation composition was relatively unchanged, the anion balance in well 41-M shifted from sulfate to bicarbonate dominance with time. Wells 10-R, 32-R and 41-R were of mixed chemical compositions. Well 10-R was basically a highly mineralized water of mixed ionic chemistry. It has remained relatively stable over time as indicated by the elemental concentrations listed in Appendix B. The water type of wells 32-R and 41-R also fell within the mix category, but both were dominated by calcium-magnesium cations and nearly equal amounts of HCO_3 and SO_4 anions.

The remainder of the wells within the study area were basically a Mg-Ca-SO_4 water type. The actual ionic concentrations of the major elements varied substantially from one well to another within the same or different aquifers (Appendix B). If the concentrations of the major contributing cations and anions were compared, expressed in mg/l, a somewhat different ion predominance would have been interpreted. It was not an uncommon occurrence in some Rosebud wells for calcium to equal or exceed magnesium values. Sodium levels exhibited incidental dominance as well. The McKay wells exhibited an overall Na cation dominance when chemical constitutents were expressed as mg/l. The ionic balance of the overburden and spoil aquifers were relatively unchanged throughout the period of study.

A possible explanation for the significant differences in Na milliequivalents may have been related to varying proportions and types of clay minerals associated with each aquifer. The coal aquifers were generally bounded by varying types and thicknesses of interbedded clay and shale layers. This could have had a definite effect upon the higher percentage of Na cations in the associated ground water as compared to the overburden aquifer. The interburden between the Rosebud and McKay seams was primarily silty clay and shale which may have accounted for the elevated Na percentages and reduced Mg concentrations observed due to the cation exchange capacity of the clays.

Mineralogy (Hydrogeochemistry)

The principal reactive minerals of interest within the sedimentary rocks comprising the spoils and overburden include dolomite, calcite, aragonite, gypsum, anhydrite, pyrite, kaolinite, illite and smectite. The mineral composition was determined through the use of core data, thin sections and X-ray diffraction techniques (Hounslow et al., 1978; Schafer et al., 1979 and Dollhopf et al., 1977b).

Carbonates, sulfates, clays and sulfides appear to have had the greatest influence upon the basic hydrochemistry of the waters analyzed. Subsurface waters in contact with sedimentary rocks derive most of their calcium from the solution of calcite, aragonite, dolomite, anhydrite and gypsum. One of the most soluble minerals in the overburden and spoils is gypsum ($CaSO_4 \cdot 2H_2O$). Even though it may often constitute <0.1% by volume, its rapid dissolution may be responsible for a major portion of the Ca and SO_4 found in spoils and ground waters. Calcite ($CaCO_3$) is readily dissolved in water if there is an abundant supply of H^+ ions. The dissociation of carbonic acid (H_2CO_3) is an important source of H^+. If there is abundant CO_2 present, the dissociation will continue only as far as the bicarbonate stage.

$$CO_2+H_2O \rightleftharpoons H_2CO_3 \rightleftharpoons H^{+}+HCO_3^{+}$$

$$CaCO_3+H^{+} \rightleftharpoons Ca^{2+}+HCO_3^{-}$$

Not only does soil air usually contain abundant CO_2 concentrations, but often large amounts of CO_2 can be attributable to the diagenesis of organic deposits (i.e., coal deposits) which enhance the solubility of Ca and HCO_3 in associated ground waters (Davis and DeWiest, 1967).

A common source of magnesium in sedimentary rocks is dolomite ($CaMg(CO_3)_2$). Calcite also commonly contains some magnesium in its composition and will yield both cations upon dissolution. Carbon dioxide also controls the solubility of $MgCO_3$, and its solubility is much greater than for $CaCO_3$. Consequently it is normally not precipitated as readily as calcite. Other magnesium salts ($MgSO_4$, MgCl) are also more soluble than their calcium counterparts.

Sodium is commonly found in natural waters as a solution product of the weathering of evaporite minerals, marine shale and Na-plagioclase. Sodium concentrations in natural waters are generally quite variable, but waters with high TDS values (1000-5000 mg/l) normally have Na levels in excess of 100 mg/l (Davis and DeWiest, 1967).

The bicarbonate and carbonate ion content in water is generally a measure of the buffering capacity. Common sources of these anions in ground water are the solution of carbonate rocks (dolomite, limestone, etc.), diagenesis of CO_2 from organic compounds and CO_2 from the atmosphere and soil. The presence of HCO_3 versus CO_3 is usually determined by the pH. Above a pH of 8.2, HCO_3 ions dissociate to CO_3 ions. Below a pH of 4.5, most of the HCO_3 ions are converted to carbonic acid (H_2CO_3). In between these two values, carbonate ions will normally add H+ ions to become HCO_3 ions.

The sedimentary rocks in association with coal beds (i.e. carbonaceous shales), will often yield substantial amounts of sulfate ions through the oxidation of pyrite and/or marcasite (FeS_2).

$$2FeS_2+15/2\ O_2+4H_2O \rightleftharpoons Fe_2O_3+4H_2SO_4$$

Most sulfate compounds are usually quite soluble in water. Sulfuric acid produced in the oxidation of pyrite and marcasite is commonly dissociated and buffered by the bicarbonate and carbonate ions dissolved from the associated carbonate rocks, thus liberating the SO_4 ions into solution.

TDS

To determine the relative degree of mineralization between aquifers, mean TDS values for all samples within each aquifer were compared. The mean values were 1593, 1360 and 1550 mg/l for the overburden, Rosebud Coal and McKay Coal respectively, which suggested a similarity between the 3 aquifers in the degree of mineralization.

Table 5 lists the mean and range TDS values exhibited by all wells within the study area. The range of mean TDS values for wells completed within the overburden aquifer showed the broadest variability (335-3098 mg/l). This range in mean values was followed closely by the Rosebud wells (434-2988 mg/l). The McKay wells exhibited the smallest variability in mean values which ranged from 657-2651 mg/l.

The variability of TDS with time was plotted for selected wells to determine if any long-term trends were observable (Figure 10). The graphs illustrated there were numerous short-term fluctuations in TDS of relatively small amplitude. Most of the variations were of such small magnitude that one could not confidently state that major changes had occurred. Some of the small scale variations may have been attributable to possible analytical error, as there appeared to be some relationship in variation between wells of the same and different aquifers. It was quite unlikely that identical changes in the hydrochemistry would occur simultaneously in each well for each sampling period. Overburden well 32-0 exhibited a somewhat erratic

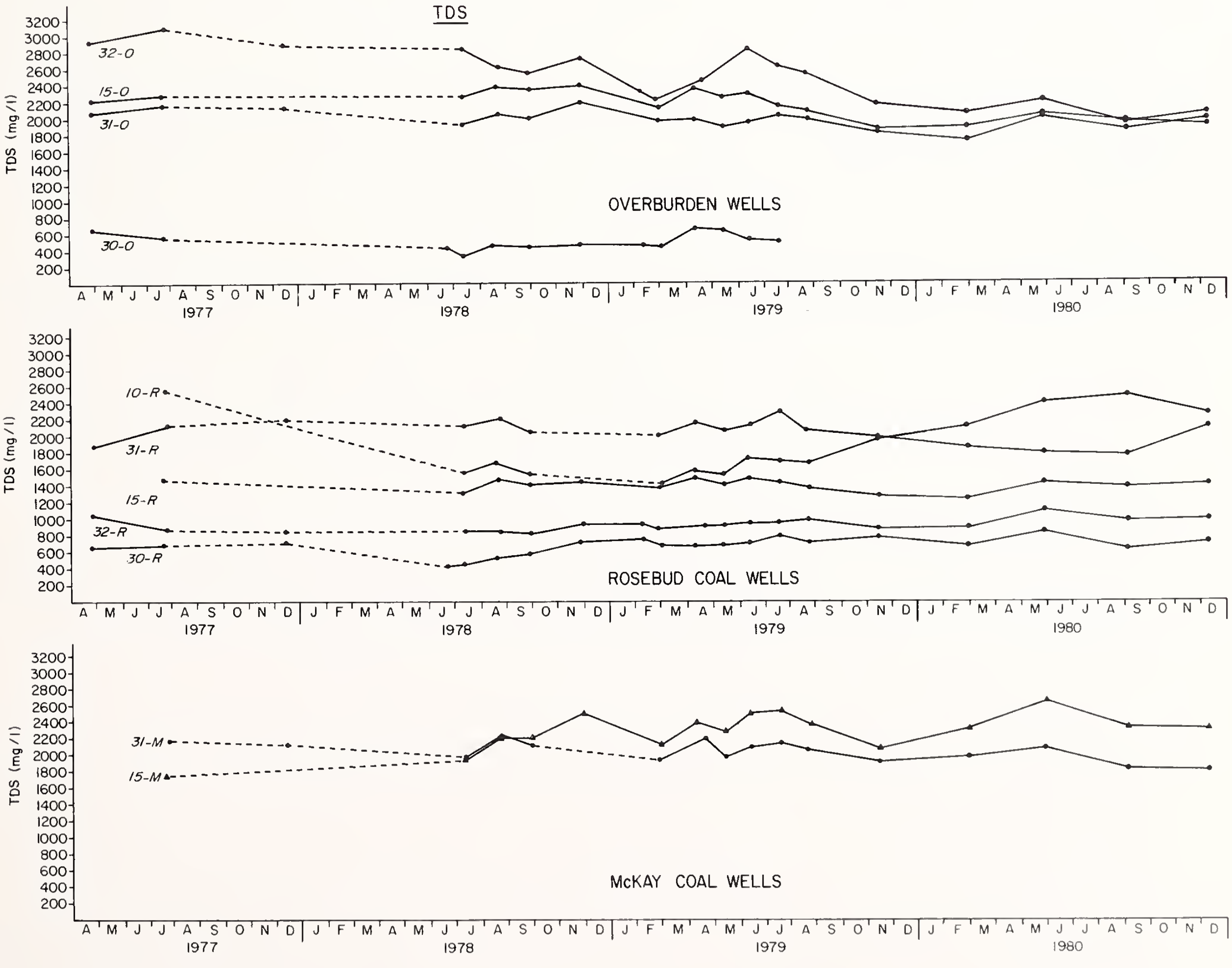


Figure 10. Variation of TDS concentration with time for selected wells in the research area, Western Energy Company Mine Area B, Colstrip, Montana.

variability with time. This may have been related to changes in the piezometric surface as a result of changes in the ground-water mound which formed beneath the surface water impoundment located adjacent to well set 32.

Extreme mineralization in spoils well CR-2 with a mean TDS of 4398 mg/l and a range of 3196-4933 mg/l, may have been attributable to the exposure of fresh mineral surfaces on unconsolidated rock fragments to the active weathering processes. Another spoils well, D-2, had a TDS mean and range substantially lower than CR-2 at 1947 mg/l, 1805-2025 mg/l, respectively. Only three repesentative samples were collected from well D-2.

The extreme salinity of the water from CR-2 would suggest substantial treatment prior to use for most irrigation purposes and its suitability as a palatable water source for livestock is questionable.

pH and Alkalinity

Aquifers were compared with one another by utilizing the mean values for pH and alkalinity from the wells of each aquifer. A mean value was determined for each aquifer by averaging all values from wells within each aquifer. A range of these mean values was also determined for each aquifer.

The comparison showed the McKay aquifer had the highest mean pH of 7.5 with a range of 6.8-11.7. The average was influenced strongly by high pH values for well 41-M. The mean pH values for the Rosebud, overburden and spoils aquifers were 7.3, 7.4 and 7.4 respectively. Generally, with the exception of certain wells, pH values varied little between aquifers.

Eight of the 15 Rosebud wells expressed mean alkalinity values between 400-600 mg/l. Three had means >600 mg/l. There were four of eight McKay wells having mean values betwen 400-600 mg/l and one >600 mg/l. The overburden wells expressed the lowest number of wells with elevated mean alkalinity values. Four of eleven wells fell between 400-600 mg/l and only one exceeded 600 mg/l. The order of average alkalinity values representative of each aquifer was as follows: McKay (512 mg/l), Rosebud (507 mg/l), overburden (465 mg/l), spoils - CR-2 (302 mg/l). Of the 35 wells sampled in the analyses, only five had mean alkalinity values in excess of 600 mg/l as $CaCO_3$. Although alkalinity values >600 mg/l may be detrimental to sensitive plants, alkalinity should be of little problem in the research area.

Hardness and SAR

The same technique for comparison of aquifers was utilized for hardness (as $CaCO_3$) and SAR as in the comparison of pH and alkalinity. An average value depicting each aquifer was again utilized. Results of the average values showed most of the ground waters to be very hard (>300 mg/l). Well CR-2 expressed the highest mean at 2331 mg/l and had a range of 1910-2958 mg/l. The other means and ranges were as follows: overburden wells (993 mg/l, 306-1787 mg/l); Rosebud wells (870 mg/l, 388-2109 mg/l); and McKay (538 mg/l, 65-1571 mg/l). Although the hardness values would probably cause some problems if utilized for industrial or domestic purposes, there would likely be few, if any, problems if it was used for irrigation or a livestock water source.

The sodium hazard (SAR concentration) for all the ground waters was categorized as low. The McKay aquifer expressed the greatest range of SAR values from its respective wells (1.5-16.8) and had an average value of 3.9. One well, 41-M, exhibited the greatest range overall at (5.1-16.8) and yet its average, 10.0 still placed it at the upper end of the low sodium harzard category (SAR of 0-10) (USDA, 1969). Remaining sodium absorption ratios were 3.3 for spoils, 1.7 for the Rosebud Coal and 1.6 for the overburden.

Minor and Trace Elements

Generally, there were relatively few minor and/or trace element concentrations represented by the ground waters that exceeded the suggested limits established by the U.S. Public Health Service for domestic water supplies. Manganese was the most common trace element to exceed the suggested limit of 0.05 mg/1. All of the wells except 28-0, 28-R, 29-0, 30-0, and 41-M expressed Mn concentrations in excess of 0.05 mg/1 in every instance. Most of the wells expressed mean values within a range of 0.10-0.30 mg/1 (Appendix B). Wells CR-2, CM-1, and D-2 expressed the highest mean values of 0.407, 0.449 and 0.359 respectively. Manganese is primarily released through the weathering of manganite ($MnO(OH)$), pryrolusite (MnO_2) and/or psilomelane ($(Ba,H2O)_2Mn_5O_{10}$) These minerals are commonly formed under highly oxidizing conditions and typically found in the clays and associated sediments of oxidized bogs, lakes and shallow marine deposits (Mason and Berry, 1968). These sedimentary deposits are associated with the fluvial system in which most of the Tongue River Member sediments were deposited.

There were only a few instances where the cadmium concentrations exceeded the suggested safe limits established for public drinking and irrigation waters (0.01 mg/1). All values which exceeded this suggested maximum Cd concentration, fell within a range of 0.01-0.02 mg/1. The following wells exhibited a Cd value within this range one time only; 31-0, 31-R, 31-M, 15-0, 41-R and CR-2. Wells 7-R, 28-R, and 29-0 also expressed one Cd value within this range, but each had only one representative sample recorded before destruction.

Three wells, (31-0, CM-1, and 31-M) exceeded the USPHS standard of 0.30 mg/1 for iron in domestic waters in several instances. The mean and range for each well was: 0.66 mg/1, 0.30-1.18 mg/1; 0.31 mg/1, .010-1.08 mg/1; and 0.19 mg/1, .015-0.82 mg/1 respectively. The remainder of the wells exhibited Fe concentrations generally below 0.10 mg/1. Most of the ground waters exhibited low levels of iron in solution due to the probable precipitation of amorphous ferric hydroxide or to reduced levels produced in some wells by the action of sulfer-reducing bacteria. No samples with excessive levels of Zn, Cu, Pb, Cl, F, and B were observed.

Summary

Numerous local variations were observed in water quality from the observation wells in the study area. Variations in concentration levels of chemical parameters rather than in major changes in chemical composition from one well to another, or from one aquifer to another were observed. Similarities of water types from different aquifers suggested substantial local mixing of ground water may occur. Local stratigraphic layers and lenses of confining clay and shale tended to locally isolate aquifers found within the area to a certain extent. The isolating layers were irregular in their areal extent, especially in the overburden, and in general, made it very difficult to predict where a particular water type would be found, and how it may have been related to a water type within close proximity.

During the period of the study, no predictable trends were observed in water quality from wells adjacent to the mined areas. The water quality from spoil materials was generally much poorer than any other water monitored in the study. Ground-water flow generally has been into the mined area, as the spoils recharge from dewatering which occurred during the mining process. When the ground-water gradient is reestablished across the spoil material and ground-water actually flows through the mined area, some degradation of water quality in discharge areas will likely occur.

The water quality of the representative surface and ground water within the study may pose a few public health problems if used as a primary domestic supply, but should pose no threat for use as a livestock water source and limited, if any, problems as an irrigation supply with proper management practices.

IV

OVERBURDEN GENESIS AND GEOCHEMICAL DEVELOPMENT

The following discussion on overburden sediment analysis provides evidence that materials at the research site were deposited by a fluvial (river) system. This evidence enabled construction of an overburden depositional model which provides partial explanation for the existence of geochemical zones unsuitable for reclamation. An understanding of the depositional environment may help in locating undesirable materials for selective placement. It may also allow collection of intensive data in specific areas of concern and thus reduce costs when characterizing overburden.

ROSEBUD OVERBURDEN SEDIMENT ANALYSIS

Sandstone

During the well monitoring installation program, 1.5 m (5 ft) composite samples were collected to check visual and geophysical logs. Samples taken within the large tabular (flat lying, rectangular to wedge shaped) sandstone unit were disaggregated and textural analyses were made using sieve and pipette procedures outlined by Folk (1974). Thin sections from a core drilled at N46,080-E57,900 were prepared and analyzed using standard petrographic microscope techniques (Figure 1). The Wentworth (1922) particle size classification was used in these sediment analyses, which differed somewhat from the USDA system (Soil Survey Staff, 1951) used for general overburden characterization (Table 6).

Table 6. A comparison of textural classification systems used at the research site.

SYSTEM OF TEXTURAL CLASSIFICATION	SIZE UNITS	BOULDER	COBBLE	VERY COARSE TO FINE GRAVEL (USDA-SCS) PEBBLE (WENTWORTH)	VERY FINE GRAVEL (USDA-SCS) GRANULE (WENTWORTH)	VERY COARSE SAND	COARSE SAND	MEDIUM SAND	FINE SAND	VERY FINE SAND	COARSE SILT	MEDIUM SILT	FINE SILT	VERY FINE SILT	CLAY
USDA-SCS	mm	>256	256-64	64-4	4-2	2-1.0	1.0-.50	.50-.25	.25-.10	.10-.05	.05-.002 (b) ← SILT →				<.002
	Phi(a)	>-8	-8 to -6	-6 to -2	-2 to -1	-1 to 0	0 to 1.0	1.0 to 2.0	1.0 to 3.3	3.3 to 4.3	4.3 to 9				>9
WENTWORTH	mm	>256	256-64	64-4	4-2	2-1	1.0-.50	.50-.25	.25-.125	.125-.0625	.0625-.031	.031-.0156	.0156 .0078	.0078-.0039	<.0039
	Phi	>-8	-8 to -6	-6 to -2	-2 to -1	-1 to 0	0 to 1.0	1.0 to 2.0	2.0 to 3.0	3.0 to 4.0	4.0 to 5.0	5.0 to 6.0	6.0 to 7.0	7.0 to 8.0	>8.0

(a) The Phi (ϕ) scale was devised by Krumbein and Pettijohn (1938). It is a logarithmic transformation of the Wentworth Grade Scale based on the negative logarithm to the base 2 of the particle diameter (mm). The values are arithmetic and more convenient for use in determining statistical parameters.

(b) For overburden analysis in this study, the silt/clay boundary was determined at .0038 mm or approximately 8ϕ.

Cumulative percent versus particle size distribution curves were plotted for each sample and parameters (mode, medium, mean, standard deviation, skewness and kurtosis) for statistical analysis were determined from the curves. Statistical data determined for samples from wells 30-R and 34-R were in general agreement with similar parameters determined by Widmayer (1977) for Tongue River Member sandstone near Decker, Montana (Table 7).

From Table 7, the following generalizations can be made. In all samples, the mode was within the fine sand range, i.e., between 2Φ (.250 mm) and 3Φ (.125 mm). Phi (Φ) is a logarithmic transformation of the Wentworth grade scale based on the negative logarithm to the base 2 of the particle diameter in mm. Median values were generally within the fine sand range, but varied from fine sand to coarse silt. An average value for all samples 2.35Φ for inclusive graphic standard deviation indicates the bulk of the samples were very poorly sorted. All samples analyzed were strongly fine-skewed and the graphic kurtosis ranged from very leptokurtic to extremely leptokurtic. In general, the sediments were fine to very fine sand, very poorly sorted, strongly fine-skewed and extremely leptokurtic; that is, the fine to very fine sand had an excess of the silt and clay fraction (<4Φ) which produced poor sorting (a well sorted sample would be one in which all particles were nearly the same size), a fine-skewed distribution and, the materials in the central portion of the distribution curve

Table 7. A comparison of statistical parameters for sandstone samples in the research area and near Decker, Montana.

PARAMETER	BRIEF DESCRIPTION SEE FOLK (1974)	WELL 30-R	WELL 34-R	WIDMAYER (1977)	VERBAL VALUES FOR STUDY AREA (FOLK, 1974)
MODE	The most frequently occuring particle diameter	2.59ϕ	2.82ϕ		Fine sand
MEDIAN	The median particle diameter by weight the diameter at the 50% mark on the cumulative curve	2.77ϕ	3.15ϕ	3.23ϕ	Fine to very fine sand
GRAPHIC MEAN	The mean particle diameter computed from Φ16 to Φ84, ie, over 68% of the curve	3.48ϕ	3.87ϕ	3.43ϕ	Very fine sand
INCLUSIVE GRAPHIC STANDARD DEVIATION	A measure of sorting over 90% of the curve	2.60ϕ	1.94ϕ	1.24ϕ	Poorly to very poorly sorted
INCLUSIVE GRAPHIC SKEWNESS	A measure of a symmetry encompassing 90% of the curve	+.76	+.70	+.48	Strongly fine-skewed
GRAPHIC KURTOSIS	A quantitative measure of the departure of a curve from a normal gaussian probability curve. A ratio between sorting in the "tails" to the sorting in the central portion	4.38	2.36	2.13 (calculated)	Very leptokurtic to extremely leptokurtic
TRANSFORMED KURTOSIS	Transformation of graphic kurtosis which, in natural sediments is strongly skewed, to a form in which a normal curve has a value of .50	.79	.69	.68	

(fine to very fine sand) were better sorted than materials on the "tails" of the curve (leptokurtic). For a more detailed description of frequency distribution terms, the reader is referred to Folk (1974).

A comparison of values with other studies of Tongue River sediments suggests the tabular sandstone in the research area was somewhat more poorly sorted with more fine grained material than Tongue River Member sandstones near Decker, Montana (Widmayer, 1977). Royse (1970) found that Tongue River sediments in western North Dakota were also poorly sorted and fine to very fine-skewed, with a markedly bimodal distribution.

Poorly sorted, fine-grained sediments with much fine material are typical of low energy environments and are commonly deposited by rivers (Reineck and Singh, 1975). Evaluation of sedimentary structures along with grain size and composition analyses leave little doubt as to the origin of the Tongue River Member sandstones in a fluvial system (Dollhopf et al., 1978b, 1979; Widmayer, 1977; Royse, 1970). These data strongly suggest point bar sequences as the primary depositional mechanism for the tabular sandstone (Dollhopf et al., 1979; Widmayer, 1977).

Sandstone Classification and Genesis

Classification of sandstones has been complicated by the multitude of classification systems proposed by numerous authors. Systems proposed by McBride (1963) and Folk (1974) were used to classify the tabular sandstone units of the research area which were either sublitharenite or lithic subarkose and chert arenite, respectively. The textural maturity of the sandstone is submature, that is, the quartz grains are angular to subrounded, are poorly sorted and the sandstone contains silt and clay.

The composition of the sandstone suggests diverse sources for this material (Table 8). The cemented aggregate (lithic) grains and pebbles of clinker and/or concretion fragments were likely derived from a nearby local source such as bank and channel erosion immediately upstream. It is unlikely that these materials were transported a great distance due to their susceptibility to abrasion. The high percentage of chert in the samples indicates a preexisting sedimentary rock source for much of the sediment. Multiple sedimentation cycles were also suggested by occasional well rounded, spherical quartz grains identified in thin sections. The occurrence of polycrystalline quartz grains, feldspars, mica, and magnetite grains suggests metamorphic or igneous rocks could have supplied some of the sediment, although, under conditions of rapid burial, these minerals are known to be durable enough to exist through multiple sedimentation cycles.

Clay Mineralogy

In conjunction with the capping demonstration (Chapter VI), X-ray defraction analyses were completed on several clay samples (Dollhopf, et al., 1977b). The clay material was located immediately above the Rosebud Coal bed and was found to be generally dominated by illite and kaolinite with lesser amounts of smectite and chlorite. Quartz was common in all samples. From stratigraphic relationships, this material would appear

Table 8. Sandstone thin section[†] mineral composition from a core drilled at coordinates N46,080-E57,900 located in Area B of the Western Energy Company Mine near Colstrip, MT.

COMPONENT	PERCENT OF COMPOSITION		DECKER STUDY	COMMENTS ON SECTIONS 1 and 2
	THIN SECTION 1 @ 60 feet	THIN SECTION 2 @ 42 feet	(WIDMAYER, 1977)	
QUARTZ	35-40	40	59 (quartz + chert)	Rounded to angular, many grains exhibit etched surfaces
CHERT	30-35	25		Most grains subrounded
CALCITE	6-7	5	13 (includes grains and cement)	Grains subangular to subrounded, some grains composed of interlocking, sparite crystals
LITHIC GRAINS	5-7(a)	5-10	10	Most grains are calcite cemented silt size quartz, (a) about 2% metamorphic?
FELDSPARS	6	5-6	10 (plagioclase)	Grains smaller than quartz, some extremely weathered. Composition includes orthoclase, oligoclase? and andesine
HEMATITE	2	5		
MICA	2	<1	4 (includes chlorite)	Most is muscovite, some biotite, some particles altered
MAGNETITE	<1	trace		Some grains altered with halo of Fe stain
CHLORITE	1	trace		Small grains, some strongly pleochroic penninite?
HORNBLENDE	-	<1	3	
MATRIX	5	5	included in calcite	Fine grained calcite cement and clay
VOIDS	4-5	4-5		Percentage of thin section area

[†] THIN SECTION 1 AND THIN SECTION 2 WERE BELOW AND ABOVE THE GROUND-WATER TABLE RESPECTIVELY

to be a flood basin deposit, produced when flood flows breached the natural levee system and entered the backswamp.

The diverse composition of the clay materials suggests this sediment was derived primarily from pre-existing sedimentary sources. Most clay minerals are moderately stable and could have been transported long distances relatively unaltered under the humid-temperate climate that likely existed during much of the Paleocene in this area (Howard, 1960).

South and south eastward paleo-current directions have been noted by several researchers working with Tongue River sediments (Widmayer, 1977; Shurr, 1972; Royse, 1970). It is likely that the regional paleo-drainage was from highland areas to the west and northwest (from initial phases of uplift and folding resulting from the Laramide Orogeny) to a marine sea (Cannonball) in what is now the Dakotas.

FLUVIAL DEPOSITIONAL MODEL

To substantiate the proposed depositional model, data from numerous drill holes were used to plot textural cross sections of a portion of the research area (Figure 11). To simplify interpretation, the top surface of the Rosebud Coal was used as the vertical datum, except where stratigraphic relationships or thinning of the coal bed indicated possible erosion of this surface. In such cases, the original estimated coal surface was used as the datum. Although much of the folding was apparently post depositional, some may have been the result of differential subsidence as the degree of folding varies vertically and laterally over short distances. During the study of Fort Union sediments in northern Colorado, Beaumont (1979) considered differential subsidence as one mechanism which stabilized meander belts, thus producing "stacked" channel deposits.

A depositional model for the research area (Figure 12) was discussed in a previous report (Dollhopf et al., 1979) in which the following description of typical fluvial deposits was presented. Reineck and Singh (1973) have grouped fluvial deposits into three groups: *channel deposits*, *bank deposits*, and *flood basin deposits*. *Channel deposits* generally include all depositional units created by the active stream channel. "Deposition on point bars is the major process of sedimentation in meandering river channels" (Reineck and Singh, 1973). In addition to point bar deposits, channel lag, channel fill, and point bar swale deposits are included in this regime. Channel lag deposits consist of residual pebbles and gravels that accumulate in deeper portions of the channel. These deposits exist as lenticular bodies that are invariably discontinuous (Reineck and Singh, 1973). Point bars are depositional features formed on the inside of river meanders and vary in size with the size of the river

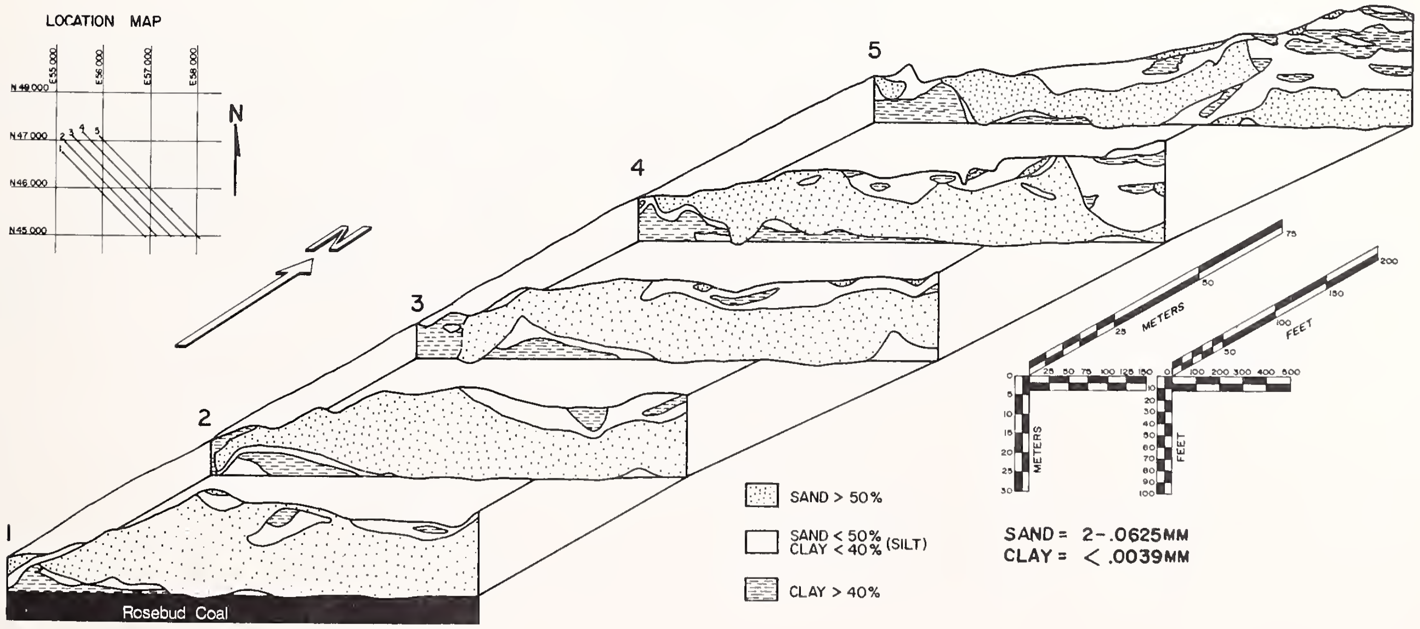


Figure 11. Textural distribution of geologic materials in Area B of the Western Energy Company Mine near Colstrip, MT.

or stream that created them. Sediments in point bars, along with the channel lag, consist of the most coarse sediment available to the stream (Reineck and Singh, 1973). Some point bar deposits in the research area vary between medium and very fine sand, in a general fining upward sequence which may be repeated. Associated with point bars are point bar swales, depressions formed between successive point bars. The swales fill with fine-grained sediments, forming small lenticular deposits. This is probably the depositional mechanism for some of the small, isolated lenses present in the research area.

Bank deposits are formed during flooding when overbank flows create natural levees and crevasse splay deposits. These sediments are generally fine sand and silts, with coarser materials deposited nearest the channel and texture becoming finer at greater distances from the channel. Much of the sandy silty material in the study area was likely deposited by this mechanism.

Flood basin deposits consist of the fine-grained silts and clays deposited when heavy flooding inundated the flood basin and its back swamps. Typical sediments are thin clay and silt stratification, marsh, and coal deposits. Most of the thick shale sequences in the research area are likely flood basin deposits.

A comparison of the model to the actual sediment distribution found in the research area suggests the bulk of the material was deposited in river channels. A large tabular sandstone unit traverses the research area in an east-northeast direction. Two other major sedimentary units include the clay and silty clay immediately above the coal, and silt and clay units above and adjacent to the sand units (Figure 11). The fine

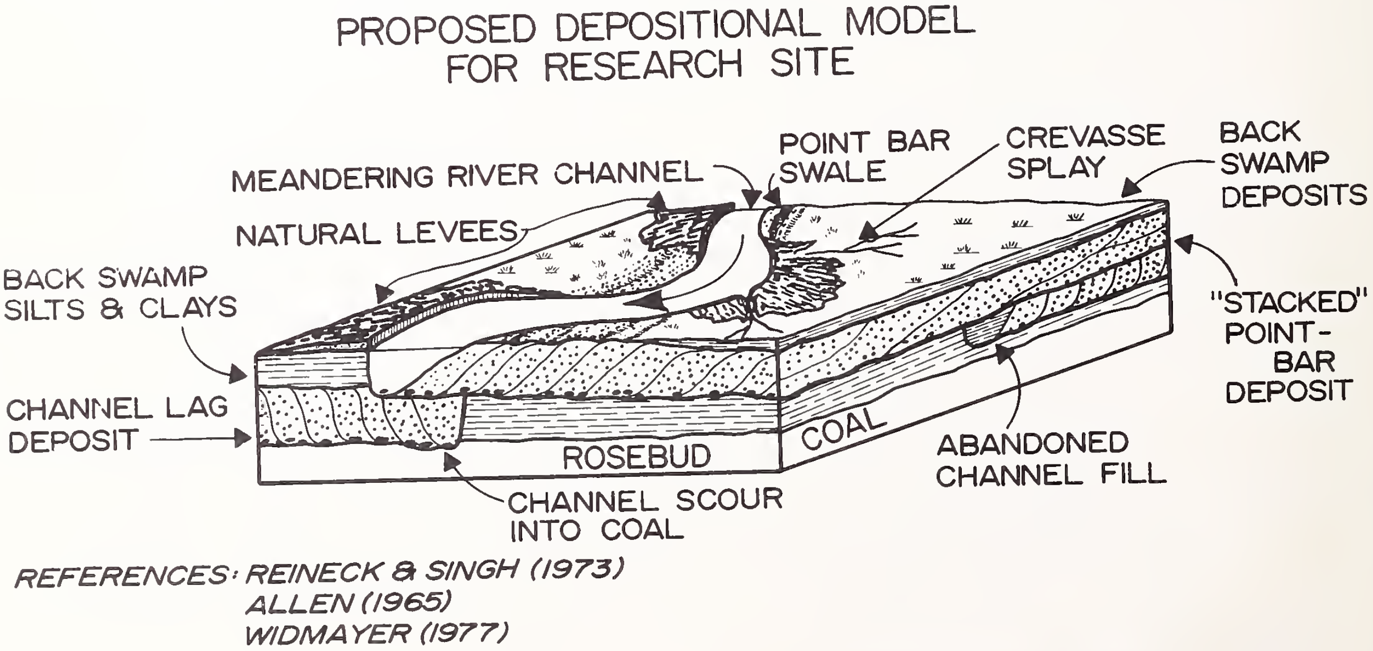


Figure 12. Depositional model for overburden in Area B of the Western Energy Company Mine near Colstrip, MT.

material immediately above the coal was most likely flood-basin backswamp sediment which was deposited primarily from suspended detritus in floodwaters. The thicker clay/silt units are likely abandoned channel fill, also composed primarily from materials contributed by floodwaters. Less carbonaceous material appears to have been associated with the abandoned channel fills and therefore, the chemical regime in which these materials were deposited likely varied from the regime associated with the backswamp deposits.

GEOCHEMISTRY

Overburden Physicochemical Characteristics

In 1976, 38 exploratory holes were drilled on a 305 m grid pattern covering an area of about 2000 hectares in Mine Area B (Figure 13). Samples collected in 1.5 m increments were analyzed for 234 different physicochemical parameters as specified by Montana Department of State Lands guidelines for overburden characterization (MDSL, 1977). Drilling and laboratory procedures are discussed in Appendix C.

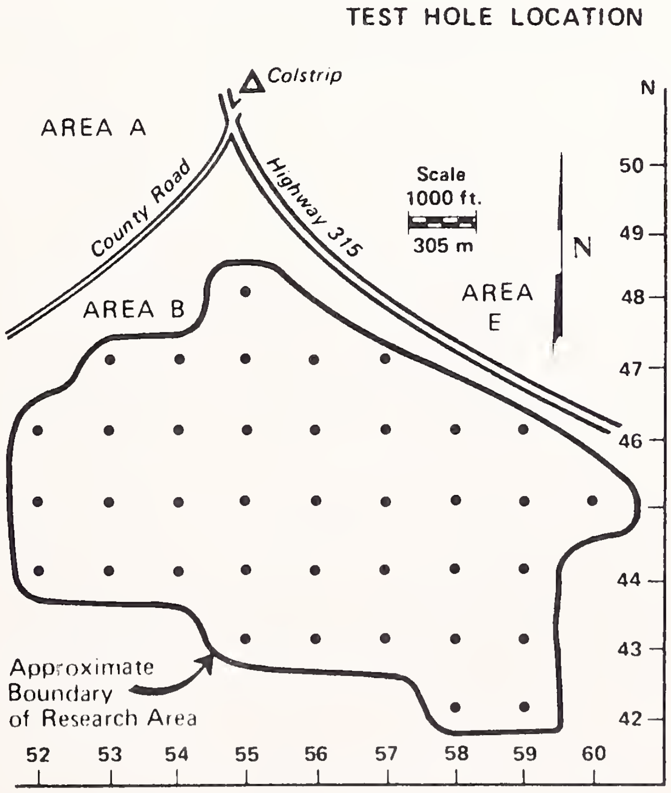


Figure 13. Location of drill holes used to characterize overburden at the Western Energy Company Mine Area B near Colstrip, MT.

Results of the laboratory chemical and mechanical analysis for Mine Area B are summarized in Table 9. It should be realized that the chemical analysis methods used were those which extract the readily soluble and exchangeable constituents in a mine soil, and do not necessarily reflect results one might obtain with total chemical analyses procedures. The procedures, then, were designed to determine the concentration of constituents that would be available to plant life. The concentration of elements readily extractable by water, i.e. ground water, could be comparatively less than values in this report. Only summary statistics will be presented here, while an earlier report presented these data in more detail (Dollhopf et al., 1978a).

The present legal criteria used for evaluating whether overburden material is suitable or unsuitable has been outlined by the Montana Department of State Lands (Table 10).

These criteria, as applied to the overburden material, are based on reviews of national literature. The literature has demonstrated or suggested that in soils having concentrations above these "suspect levels" plant production has been restrained. The literature also shows that in some cases soil concentrations above these levels resulted in degradation of associated water resources. The use of the national literature was necessary due to an

Table 9. Summary of overburden chemistry and mechanical analysis data from the Western Energy Company Mine Area B near Colstrip, Montana (see Appendix C for analysis procedures).

Analysis	Mean ± Standard Deviation	Range Min - Max	Number of Samples
pH	8.0 ± 0.4	5.0 - 9.6	887
Saturation %	36.80 ± 7.77	18.90 - 74.40	887
Conductance (mmhos/cm)	1.83 ± 1.65	0.23 - 14.80	887
Ca (meq/L)	6.26 ± 6.69	0.30 - 78.84	887
Mg (meq/L)	11.76 ± 15.54	0.30 - 118.42	887
Na (meq/L)	4.68 ± 6.17	0.04 - 60.87	887
SAR	1.48 ± 1.07	0.01 - 8.68	886
Zn (ppm)	13.44 ± 36.67	0.08 - 650.00	890
Fe (ppm)	26.43 ± 21.53	0.05 - 104.00	891
Cu (ppm)	1.37 ± 1.11	0.05 - 6.30	890
Mn (ppm)	3.18 ± 2.56	0.05 - 23.80	891
Ni (ppm)	0.83 ± 0.73	0.05 - 5.21	893
Cd (ppm)	0.15 ± 0.30	0.05 - 3.75	893
Pb (ppm)	2.80 ± 2.17	0.10 - 11.90	893
Hg (ppm)	0.026 ± 0.269	0.0005 - 0.278	893
Se (ppm)	0.013 ± 0.007	0.005 - 0.110	891
Mo (ppm)	0.863 ± 0.687	0.005 - 4.65	893
B (ppm)	0.460 ± 0.507	0.005 - 3.95	879
PO_4 (ppm)	2.65 ± 4.69	0.10 - 38.00	810
NO_3 (ppm)	0.67 ± 1.06	0.05 - 10.60	874
NH_4 (ppm)	13.77 ± 9.91	0.50 - 80.02	892
Sand (%)	44.71 ± 22.35	3.60 - 92.40	826
Silt (%)	29.53 ± 14.77	1.00 - 83.00	824
Clay (%)	25.72 ± 13.85	1.80 - 77.60	825

Table 10. Montana Department of State Lands guidelines for suspect levels in overburden material.

Analysis	Susepct Level
Conductance	>4-6 mmhos/cm
Sodium adsorption ratio	>12
Mechanical Analysis	Clay >40% Sand >70%
Saturation %	None
pH	>8.8-9.0
PO_4-P	None
NO_3-N	>10-20 ppm
NH_4-N	>10-20 ppm
Cd	>0.1-1.0 ppm
Cu	>40 ppm
Fe	Unknown
Pb	pH <6, >10-15 ppm pH >6, >15-20 ppm
Mn	>60 ppm
Hg	>0.4-0.5 ppm
Se	>2.0 ppm
Mo	>0.3 ppm
B	>8.0 ppm
Zn	>30-40 ppm
Ni	DTPA extraction, >1.0 ppm Acid Extraction, >5.0 ppm

absence or shortage of this type of information for the Fort Union Coal region. Since in many cases, these chemical studies were based on conditions different from those normally found in Montana, their applicability to any one specific region or mine site is questionable. There are exceptions to this statement for such well known parameters as sodium adsorption ratios, conductance, and nitrate; however, in general, the limits are somewhat arbitrary, especially with regard to the trace elements.

A comparison of the analysis summary (Table 9) with the suspect levels (Table 10) indicated the overburden did not contain any excessive concentrations of copper, lead, manganese, mercury, selenium, or boron. Sodium was not a problem since its concentration was relatively low and the maximum sodium adsorption ratio (SAR) was only 8.68. Although the concentration of nitrate-nitrogen (NO_3-N) was very low, mean of 0.67 ppm, a few samples exceeded the 10.0 ppm suspect level.

Several parameters exceeded the suspect level including the trace elements nickel, cadmium, zinc, and molybdenum. Levels of pH, conductance, ammonium, sand, and clay were also found to exceed suspect levels in this overburden material. The statistical mean for both ammonium and molybdenum exceeded the suspect level indicating a considerable volume of overburden in Area B may be adversely affected by these elements. The statistical mean for zinc, cadmium, and nickel plus one standard deviation exceeded the respective suspect levels. This indicated a considerable number of overburden samples contained excessive concentrations of these elements. Interpretation of cadmium results must be qualified. The minimum analytical sensitivity for cadmium was 0.10 ppm during this laboratory work. Therefore, cadmium concentration results reported by the lab as something less than 0.10 ppm were for statistical purposes divided in half. Thus interpretations regarding cadmium will include this error.

According to Montana Department of State Lands criteria the research area contained a number of chemical and physical characteristics that could be harmful to the area water and plant resources. Briefly, the following statements will allude to the environmental consequences if the plant and water resources are subjected to such excessive levels. High conductance values indicate excessive salt concentrations. If this soil is used as a root growth medium, the plant will be required to expend a greater amount of metabolic energy to take up water, thus adversely influencing the plant-water relationship. Excessively high concentrations of trace elements (Ni, Cd, Zn, Mo) can cause a direct metabolic interference in plant cell physiology. A soil with greater than 40% clay will likely have characteristics unfavorable for root penetration and water infiltration. A soil with greater than 70% sand will likely transmit water fast thus limiting water retention for plant use. Also, such a soil has little capacity for plant nutrient storage. Excessively high ammonium (NH_4) values should have a positive influence on plant production. If placed in the root zone, NH_4 will partially oxidize to nitrate (NO_3). Plant life can utilize both these forms of nitrogen in substantial amounts. The concern about NH_4, however, is its association with ground water.

The concern between ground water and overburden chemistry stems from the belief that ground water will take on, to a certain extent, the chemical characteristics of its flow medium. The development of excessive salt or trace elements in ground water may reduce the desirability of the aquifer for domestic purposes, or if unknowingly used, these waters may create metabolic problems in animal life. The specific concern with high NH_4 levels in overburden stems from its potential oxidation transformation to NO_3 during the mining process which subjects overburden in a chemically reduced state to oxygen in the earth's atmosphere. Thus a relatively sudden source of NO_3 may be made available to the areas water resources.

Overburden Particle Size Relationship To Trace Metals

One of the major concerns associated with selective placement of overburden is the expense to attain adequate characterization of overburden materials. The development of relationships between specific chemical parameters and the materials in which they occur could serve as a guide to reduce the number of boreholes, drilling footage, and chemical analyses needed to characterize the overburden.

It is known that each depositional environment is characterized by a unique geochemical regime and a unique particle size range, the latter determined by the transport energy available to the system. Therefore, a determination of the geometry of deposits produced in various depositional environments and the relationship developed between these deposits and geochemical distribution may allow a gross delineation of zones unsuitable for reclamation with relatively few boreholes.

The distribution and concentration of the trace metals Ni, Pb, and Zn were plotted against their occurrence in materials composed of sand, silt and clay. Problems encountered with drilling grease contamination made conclusions with respect to Zn impossible (Dollhopf et al., 1979). Although some Pb contamination may have also occurred, the bulk of the samples used in this analysis were obtained using a drilling rig with 15 m drill stems and grease with little Pb additive. The combination of few drill stem changes and low Pb content grease probably produced little effect on the apparent distribution of Pb reported here, but the data should be viewed with caution.

In a comparison of Ni concentration with varying percentages of sand (Figure 14a), the Ni distribution was apparently bimodal. That is, numerous samples containing low concentrations of Ni (<.5 ppm) occurred within the sand composition range of 12-22% and 70-84% sand. At Ni concentrations of 1.0 to 1.5 ppm, samples were more evenly distributed over the entire range of sand composition. The plot of Ni distribution with clay content exhibited a marked number of samples of low level Ni in the 8-16% clay range (Figure 14b). Of greater importance however, was the wider distribution of samples with Ni levels >1.0 ppm, indicating that at higher concentrations, Ni had little correlation with textural composition.

Of the total sample population (3919), 2.7% exceeded the DTPA extractable suspect Ni concentration of 1 ppm. Of the 2.7% suspect samples, 0.84% occurred in the ≥50% sand composition and 1.84% in the <50% sand composition. Hence, for the research site, suspect levels of Ni >1 ppm were over twice as likely to occur in materials composed of <50% sand as compared to materials of ≥50% sand. When the overburden clay content is >40% and when Ni levels are >1 ppm, material is considered unsuitable for reclamation in Montana. When Ni levels exceeded 1 ppm, 44.2% occurred in overburden with ≥40% clay and 55.8% fell in the <40% clay range. This suggests determination of clay content alone cannot be used to identify occurrences of Ni in the >1 ppm range.

The mean textural values of all samples containing suspect Ni concentrations (>1 ppm) was 35% sand, 30% silt and 35% clay. Material with this composition would generally be associated with crevasse-splay, flood basin or abandoned channel fill deposits.

In similar analyses, Pb was found at low concentration (< 1 ppm) in the overburden at both low and high sand contents (Figure 14c). Of all occurrences of detectable Pb, 47% occurred in materials with ≥50% and 53% occurred in materials with <50% sand. However, at Pb concentrations >2 ppm, there was a marked increase of Pb occurrences in the 10-30% sand range and in materials with clay >40% (Figure 14c and 14d). For example, while no Pb occurrences at concentrations of 4.5 ppm were found in materials composed of ≥50% sand, over 2% of the total number of recorded Pb occurrences (3570) were found in materials with less than 50% sand. This trend suggests that overburden with Pb concentrations unsuitable for reclamation (>15 ppm) would be most often associated with materials having high clay percentages, that is, with backswamp or channel fill deposits.

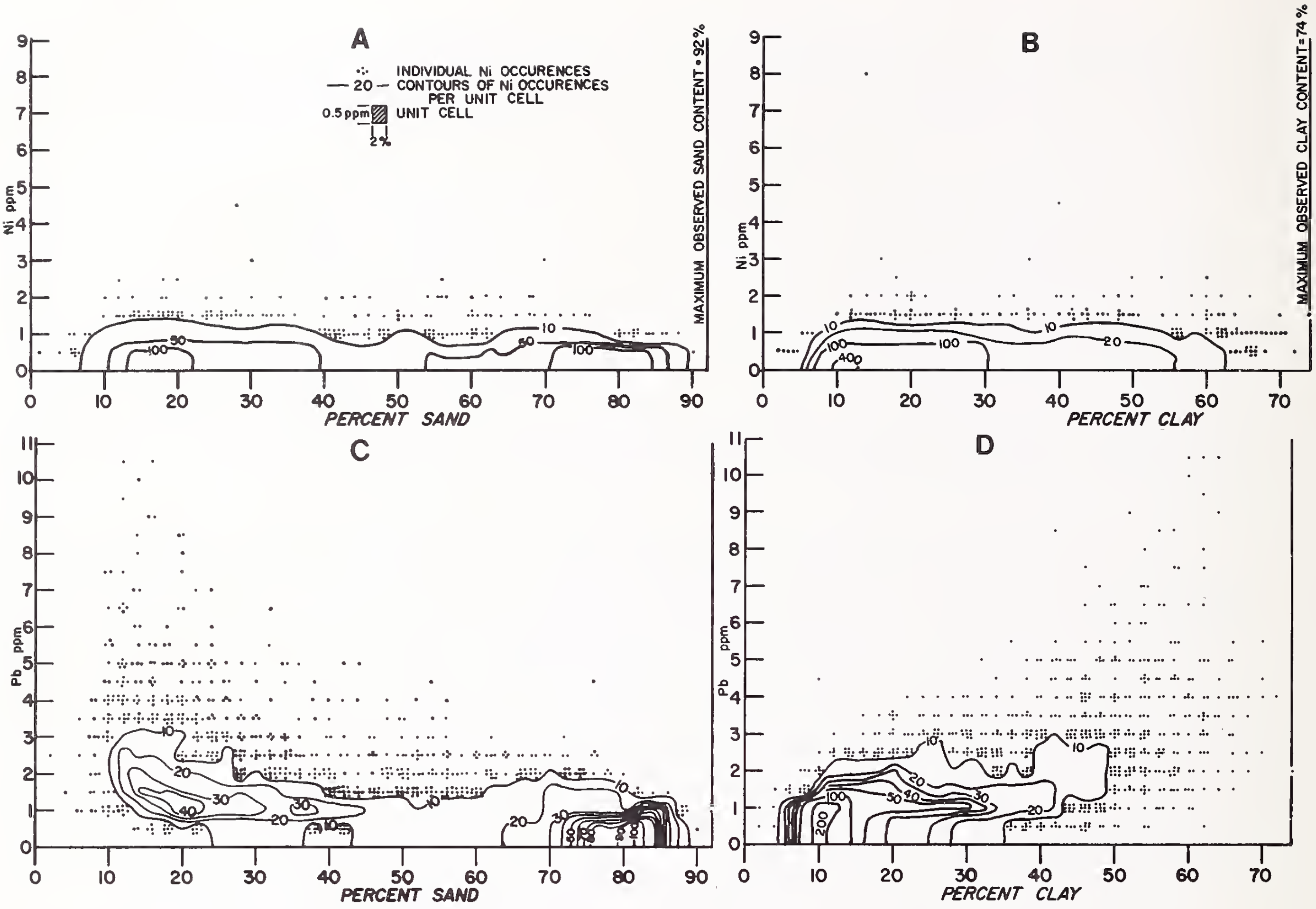


Figure 14. Distribution of Ni and Pb within overburden sand and clay percentages in Area B of the Western Energy Mine near Colstrip, Mt.

Little is known about the chemical forms in which many trace elements exist. Neither DTPA extractions or total analyses yield information about mineral composition, thus the following statements are somewhat speculative. A review of sandstone mineral composition (Table 8) suggests two possible mineral sources for the Ni observed in sandstone. Chlorite and magnetite both generally contain a percentage of Ni in substitution for Fe in the crystal structure. Upon weathering, these minerals may release small quantities of Ni which would be chelated by DTPA extraction. Chlorite is also a major constituent of clay size particles (Table 8), and may contribute a small amount of the extractable Ni observed.

Of the mineral composition found in sandstone, none are known to be significantly lead bearing. Thus, much of the DTPA extractable lead is likely adsorbed on clay minerals. Some quantity of clay was found even in samples containing the highest percentage of sand.

The differences between distribution patterns observed for Ni and Pb are likely similar to other trace element distributions. That is, each trace element may show a unique distribution within various textured materials. If, as in this study, suspect Ni levels were encountered, identification of depositional zones may outline areas in which suspect Ni levels are more likely to occur. However, the wide distribution of suspect Ni with varying textural composition found at the research site would still require intensive drilling to identify all suspect Ni zones. Conversely, the trend for higher concentrations of Pb to occur in materials with high clay content may allow a nearly complete delineation of suspect Pb zones by identification of depositional system trends and thereby greatly reduce costs for analyses.

Detecting Unsuitable Saline Overburden Material

An analysis of materials with excess soluble salt levels, (EC >4 mmhos/cm) was used to determine to what depth drill holes would have to be drilled to reliably locate these materials. Of 508 boreholes, 150 boreholes at the research site encountered material with EC values >4 mmhos/cm. A plot of hole depth against successful identification of excess EC suggested that EC distribution at the research site was a surface-related phenomena (Figure 15). Of the 150 boreholes containing EC zones >4 mmhos/cm, 73 holes (55%) were identified by the first 1.5 m (5 ft) sample. The second 1.5 m sample identified an additional 28% of the holes with excess salt. Drilling to a depth 6 m (20 ft) identified all but 2% of the holes with EC zones >4 mmhos/cm. This suggests that at the research site, there is little need to analyze samples for EC below the 6 m depth. The distribution of zones with high EC values is likely very site specific and is dependent on the climate, topography and the geologic stratigraphy of the site. Therefore, it will probably be necessary to determine the validity of this relationship at each site.

It is likely that the relationships of most chemical parameters with geologic materials can be clearly established. Although these relationships may not delineate all suspect zones, they should significantly increase the probability of locating suspect zones with fewer analyses and perhaps fewer boreholes.

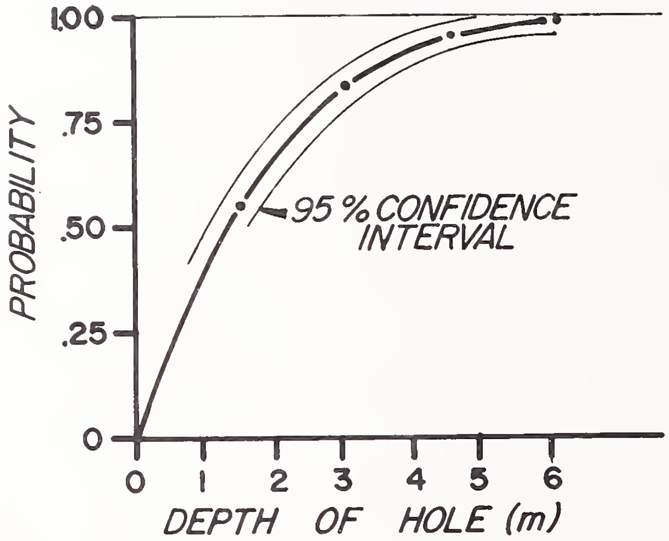


Figure 15. The probability of correct identification of all boreholes containing material with EC >4/mmhos/cm versus depth of sampling in Area B at the Western Energy Company Mine near Colstrip, MT.

V

OPTIMUM OVERBURDEN DRILLING PATTERN

The Surface Mining Control and Reclamation Act (PL 95-98) repeatedly indicates unsuitable (toxic) overburden materials must be special handled so as not to limit reclamation of the plant, soil and water resources. Accurate delineation of inhibitory overburden zones at a mine site is a technological problem.

Current technology provides for drilling of boreholes through the overburden for chemical analysis of various parameters at period depths. Western states such as Montana, Wyoming and North Dakota are required to sample overburden with boreholes spaced 600 to 1500 m apart. With these sampling intensities, a mine may move overburden for several years based on physicochemical results from only 2 or 3 boreholes. Overburden sampling guidelines are not so stringent in Colorado, New Mexico, Arizona and Utah. Work by the U.S. Geological Survey (Hinkely et al., 1978) indicated that almost as much information about overburden chemistry can be obtained from a single hole drilled anywhere on the mine site, as compared to information from many holes. This result was true when boreholes were spaced greater than 600 m apart. The stratigraphy and geochemistry were so variable between boreholes that the lateral and vertical extent of unsuitable overburden materials could not be determined. Therefore, it may be a futile effort to plan selective overburden handling operations since accurate extrapolation of physiochemical characteristics between boreholes 600 to 1500 m paart is not possible.

The goal of this study was to determine how intense overburden needs to be sampled in order to accurately characterize the extent of materials unsuitable for reclamation.

METHODS

At the Western Energy Company Mine Area B near Colstrip, MT., 270 sites were drilled within a 50 ha area (Figure 16). Samples were collected in 1.5 m depth increments and analyzed for EC, Ni, Cu, and particle size distribution (Appendix D). A Bucyrus Erie 60-R and a Portadrill 524 were both used to construct boreholes using the air rotary method without aid of solutions.

Based on the physicochemical analysis of each sample taken, the following question was considered. If the drilling location sampled was inhibitory (or noninhibitory) in some element, could one predict that the overburden was similarly inhibitory (or noninhibitory) within certain distances from the predicting sample? If the answer to this question was yes, then it would seem reasonable to expect that the inhibitory status of the sampled site should accurately predict (or agree with) the inhibitory status of samples taken within these regions. Furthermore, it might be expected, that for regions close to the location of the predicting sample, that this accuracy should be better than for those regions further away.

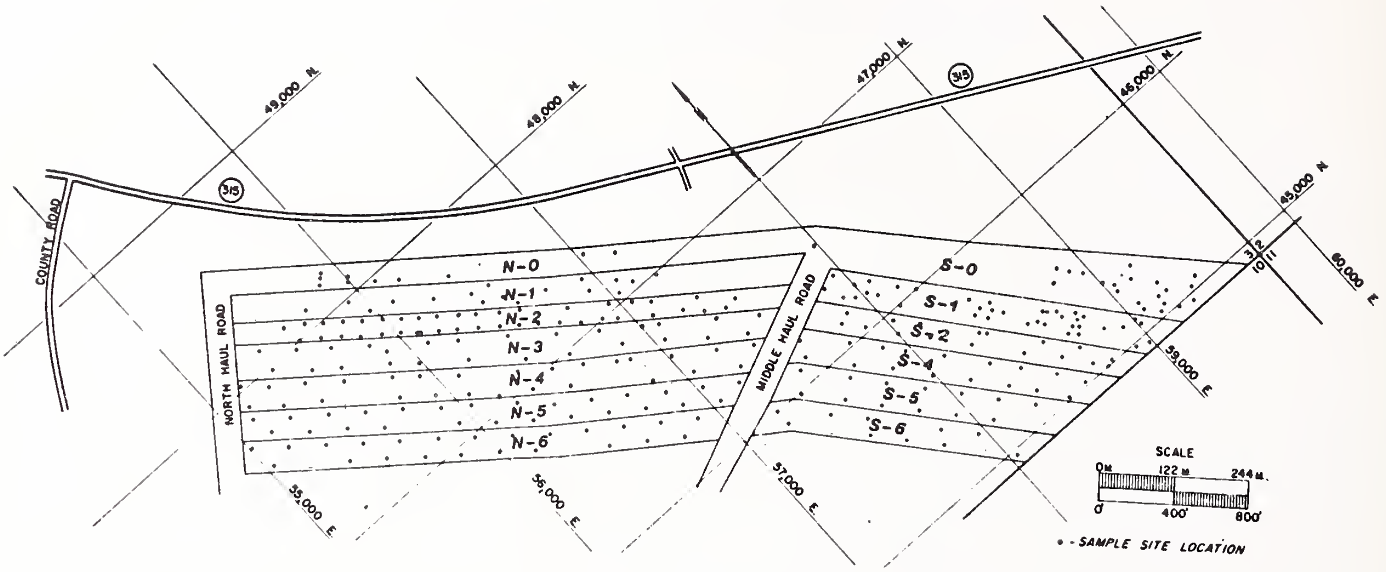


Figure 16. Aerial view of the Western Energy Company Area B, near Colstrip, MT., showing locations of boreholes used in drill site intensity evaluation.

To help document these possibilities, eight regions were selected in relation to each sample drill site location. They consisted of areas located 0-76 m, 76-152 m, 152-228 m, 228-304 m, 304-380 m, 380-456 m, 456-532 m, and 532-608 m from a central predicting drill site.

For each region, the total number of accurate predictions made by the inhibitory status of the central predicting drill sites on those drill sites located within the region were tallied along with the total number of incorrect predictions. The proportion of accurate predictions for a region would then be a measure of how well the inhibitory status of a central drill site could characterize the overburden in adjacent areas. A computer program was developed to perform this analysis. Spacial relationships between drill sites were digitized into the computer program using a graphics tablet and coordinate system. The statistical model has been discussed in more detail elsewhere (Dollhopf et al., 1978 b).

OVERBURDEN LOCATION OF MATERIALS UNSUITABLE FOR RECLAMATION

Before discussing optimum drilling intensity results, a basic knowledge of the lateral and vertical extent of unsuitable overburden material will be helpful. Using physicochemical results from all 270 boreholes on approximately a 60 m sampling grid within the first seven mine passes, the location of materials unsuitable for reclamation were delineated. The distribution of soluble salts in excess of 4 mmhos/cm is shown as a surficially located 4 to 5 m thick mostly continuous deposit winding through the mined area (Figure 17). In the foreground the saline zone cropped out at the surface (topsoil was already removed) but generally remained below the surface towards the rear of the diagram. At

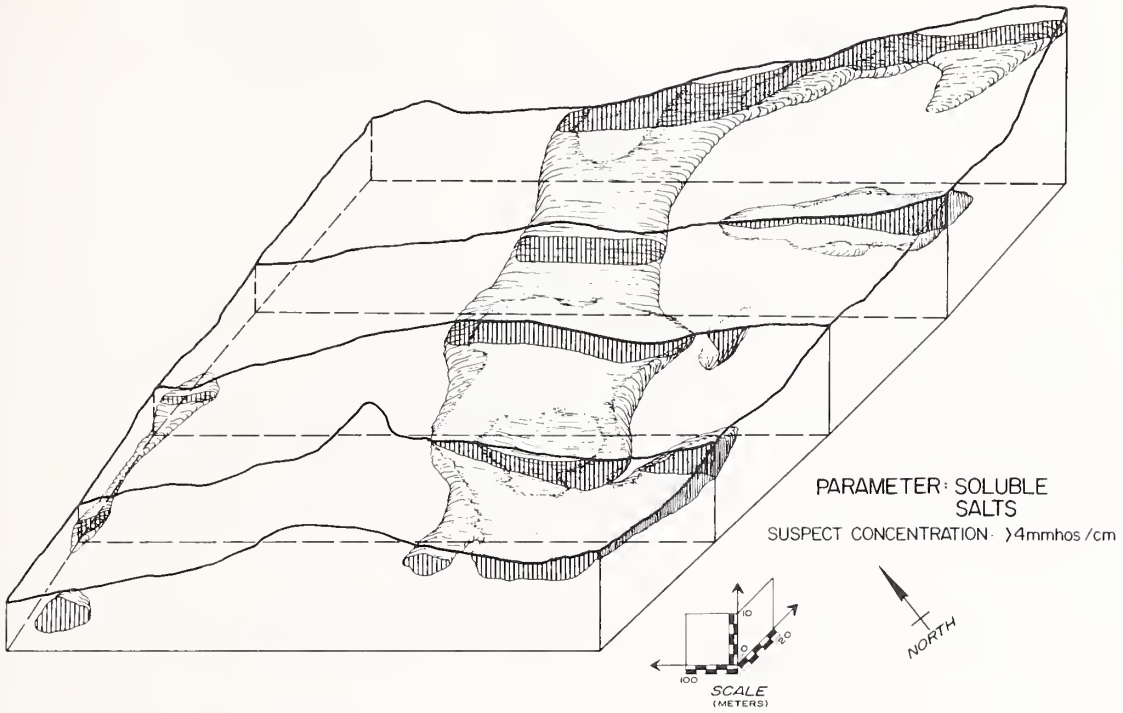


Figure 17. Overburden fence diagram showing location of saline material in eight mine passes (see Figure 16) at the Western Energy Company Mine Area B, Colstrip, MT.

this site, the mine operator would have been required by regulation to characterize the area with as few as two or three boreholes. With only 3 boreholes (600 m grid) the unsuitable material could have been entirely undetected. If detected, the true lateral and vertical extent could not have been mapped with three boreholes. Similar fence diagrams constructed for inhibitory levels of sand, clay and nickel also demonstrated that intensive overburden sampling would be required to delineate these materials (Figures 18, 19, 20, and 21). All suspect concentrations are those presently being used to evaluate overburden in Montana. Large deposits of clay immediately above the coal seam were present in shallow overburden (10 to 15 m), and extensive sandstone units were present throughout the mine area. Overburden materials with elevated levels of Zn found immediately above the coal seam were small in dimension and would probably dilute to acceptable concentrations during the material handling process. Most deposits of elevated Ni concentrations adjacent to the coal seam were also small in dimension, but one large deposit probably would not have diluted to acceptable levels in the resultant spoils.

The genesis and geochemical development of these zones of unsuitable materials was discussed earlier in Chapter IV.

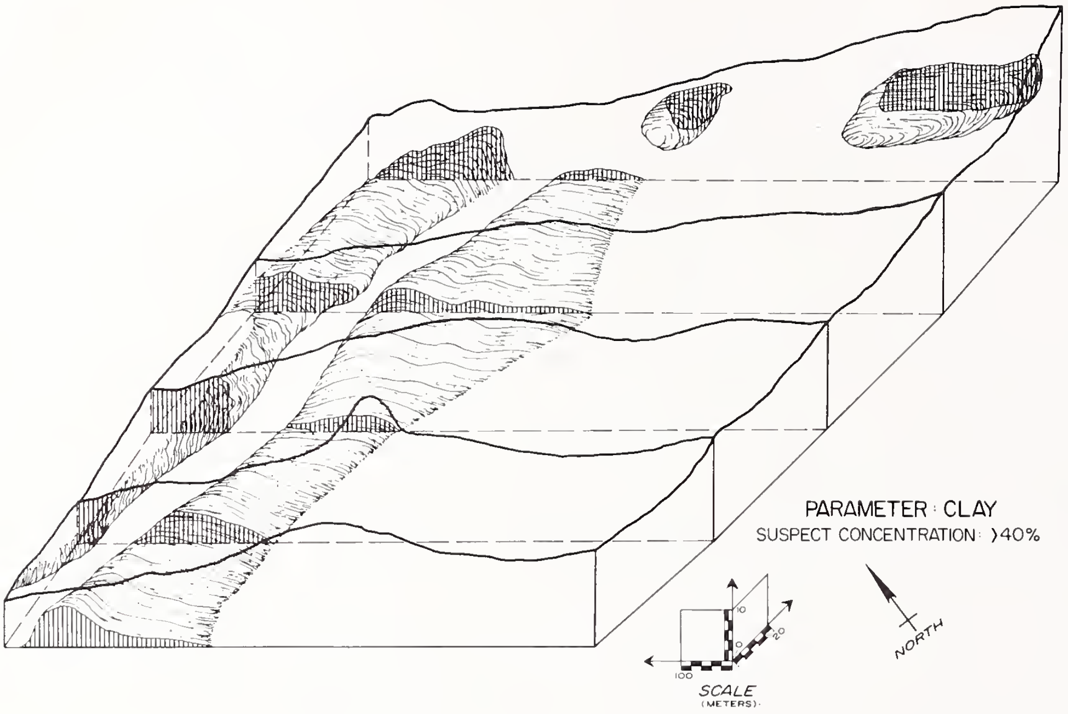


Figure 18. Overburden fence diagram showing location of clay materials unsuitable for reclamation in eight mine passes (see Figure 16) at the Western Energy Company Mine Area B, Colstrip, MT.

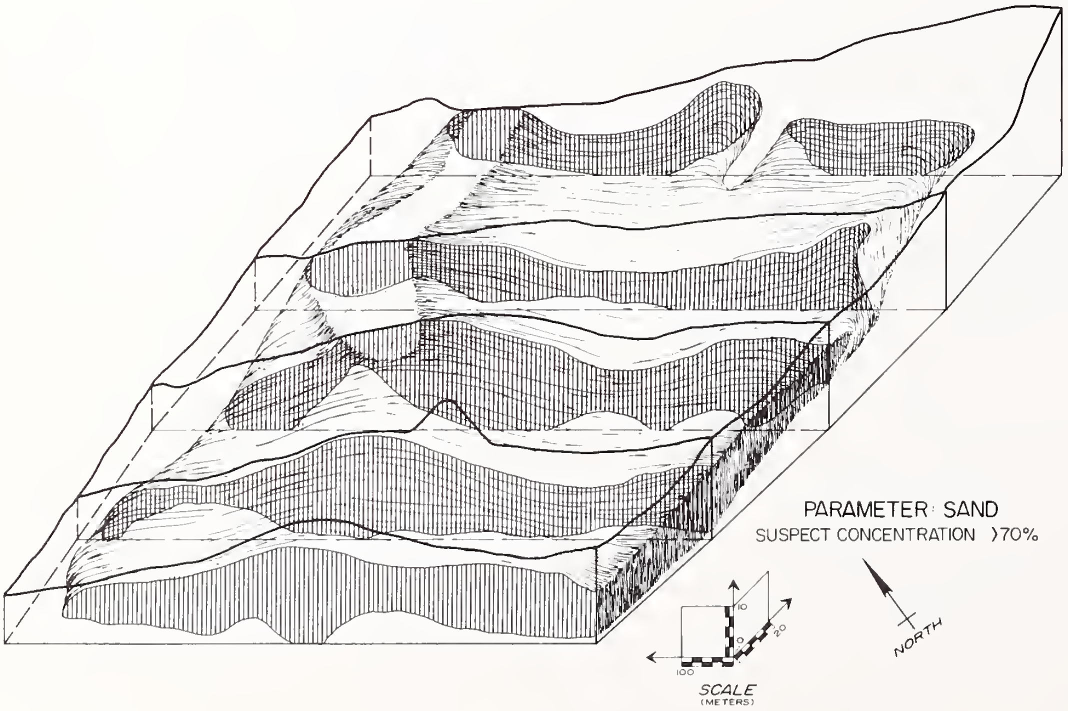


Figure 19. Overburden fence diagram showing location of sand materials unsuitable for reclamation in eight mine passes (see Figure 16) at the Western Energy Company Mine Area B, Colstrip, MT.

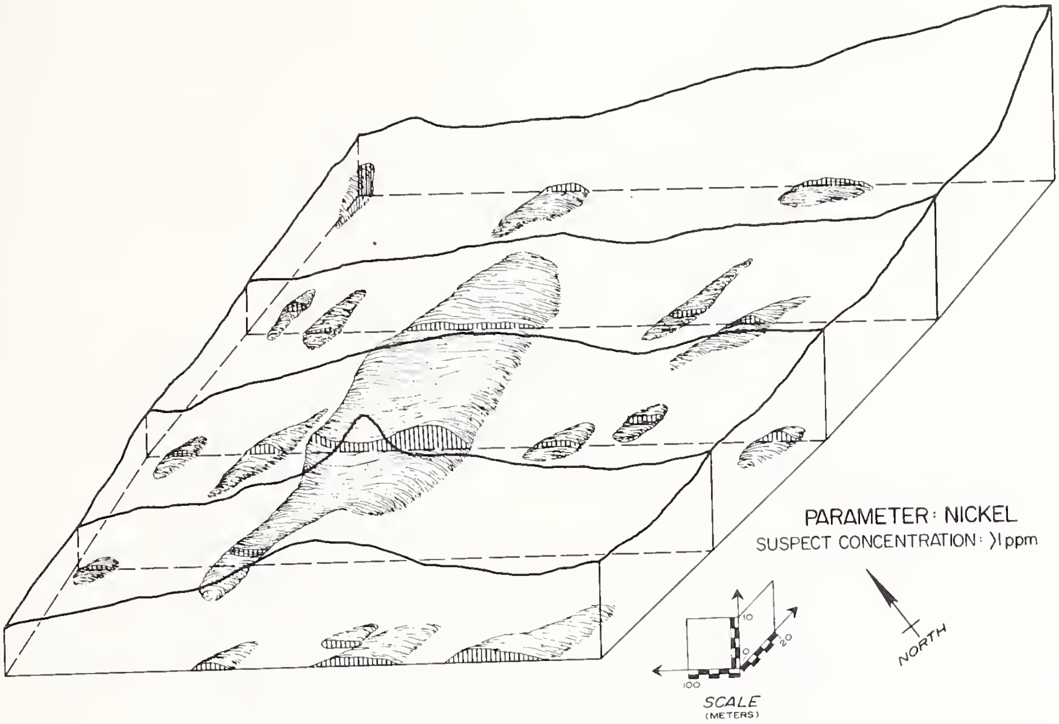


Figure 20. Overburden fence diagram showing location of material with Ni concentrations unsuitable for reclamation in eight mine passes (see Figure 16) at the Western Energy Company Mine Area B, Colstrip, MT.

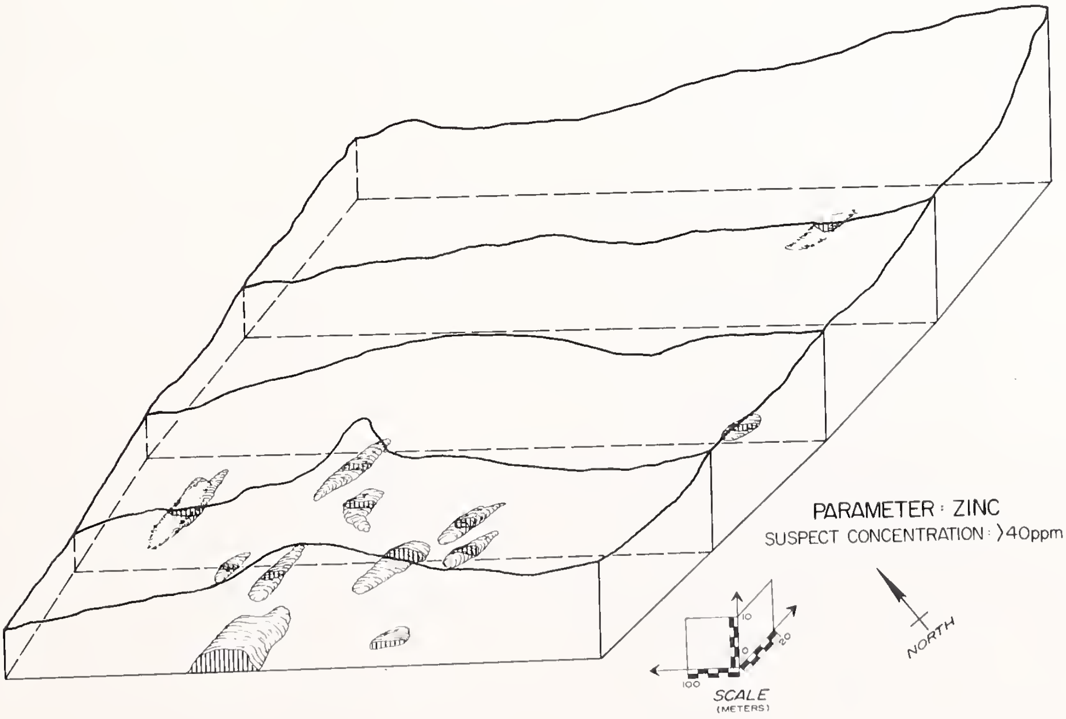


Figure 21. Overburden fence diagram showing locations of material with Zn concentrations unsuitable for reclamation in eight mine passes (see Figure 16) at the Western Energy Company Mine Area B, Colstrip, MT.

ACCURACY OF DIFFERENT DRILLING INTENSITIES

The statistical techniques presented in the methodology section are applied to overburden properties involving soluble salts, clay, and nickel. Problem levels that denote inhibitory material in these properties include, respectively, those values that exceed 4 mmhos/cm, 40%, and 1.0 ppm (MDSL, 1977).

Of course, there is one inescapable conclusion. That is, as drilling intensity increases, our ability to accurately define overburden inhibitory zones increases (Figure 22). Ideally, in biological-environmental systems we would like to make correct evaluations at least nine times out of ten, i.e. with 90% accuracy. If we apply this criterion to the regression equations presented in Figure 22 to clay and soluble salt prediciton then we see the required drilling intensity would approach a 430 m grid. Our ability to characterize unsuitable overburden with drill holes 600 to 1500 m apart as recommended by States, has an accuracy of only 45 to 60%. This means that it is probable that a typical premine overburden assessment program missed detecting half of the unsuitable materials in the project area. Since our ability to predict inhibitory (or noninhibitory) status of overburden between two boreholes approaches a 50% accuracy, our ability to predict the presence or absence of inhibitory material between such boreholes is very limited. Therefore, when reconnaissance overburden sampling is performed it may very limited. Therefore, when reconnaissance overburden sampling is performed it may very likely be a false interpretation to assume noninhibitory overburden will be present between two adjacent boreholes which did not intercept inhibitory material. Considering that western states sample overburden in a 600 to 1500 m grid fashion, it is likely that incorrect interpretations are being made resulting in inhibitory materials being unknowingly deposited either in the future aquifer zone or root zone.

Note also that since all three curves in Figure 22 approach a plateau or asympote on or before a 600 m sampling intensity, these data are in close agreement with the USGS study discussed earlier (Hinkley et al., 1978). That is, our ability to characterize overburden in a mine project area with one borehole, e.g. an 8 km (5 mi) grid, is in effect no worse than with boreholes on a 600 m grid since we are still delineating inhibitory materials with only about a 50% accuracy.

Sampling the overburden to assure all unsuitable materials have been delineated (90% accuracy) would mean costs associated with overburden characterization would skyrocket into millions of dollars compared to several hundred thousand dollars for today's typical Western strip mine operation. Such costs may represent a financial burden to the mining industry. Sampling the overburden in a two step fashion may provide a desirable compromise between accuracy and costs. For example, the overburden at a mine site could first be sampled on a 600 m grid. This would not provide good accuracy for delineating the location and extent of inhibitory materials, but such a sampling scheme would very likely identify which overburden chemical and physical parameters are present at inhibitory levels at the mine site. Then the entire area would be drilled intensively to attain a characterization

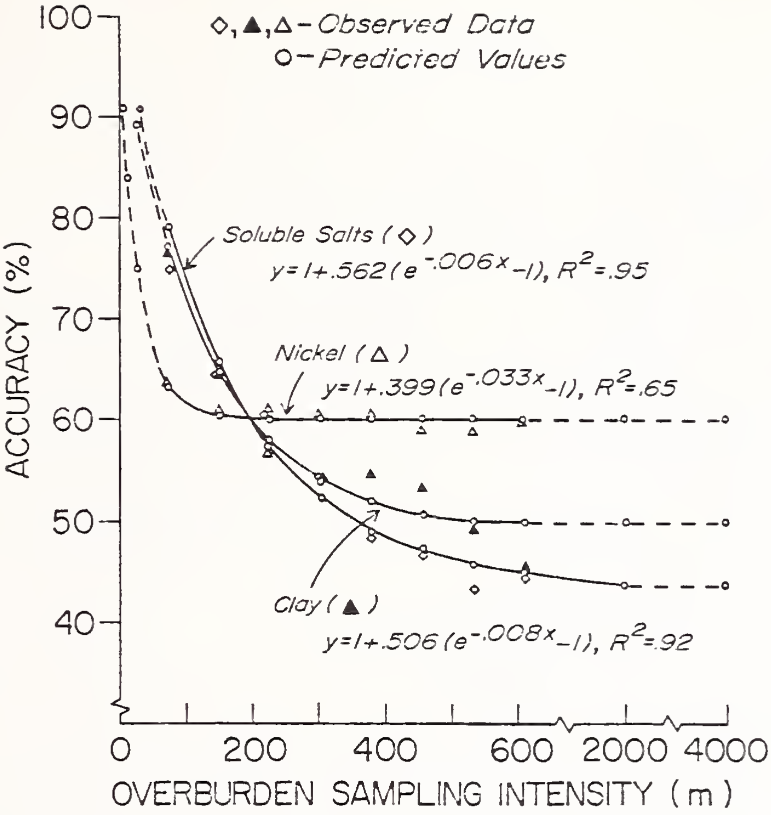


Figure 22. Accuracy in characterizing unsuitable overburden zones as a function of drilling intensity for the parameters soluble salts, nickel, and clay.

accuracy of 80 to 90%, i.e. a 60 m grid, but these samples only need to be analyzed for those parameters which were present at inhibitory levels during the initial reconnaissance. Given any overburden characterization study, approximately 15% of the cost will be for drilling services, the remaining cost will be largely sample analysis cost. Thus, a two-fold approach as described above could decrease overburden characterization costs substantially.

A recent study assessed the feasibility of developing a toxic overburden detection system (Bruhn et al., 1980). They found that the technology was not present for a detection system that operated from the surface, but indicated certain downhole probes utilizing physical chemistry approaches were feasible. Development of a downhole probe that incorporates natural gamma, neutron activation, gamma-gamma, temperature, caliper, induction, pH and spectral should when fully refined be capable of providing immediate on-site delination of unsuitable overburden materials. Borehole drilling costs would still be present, but the cost of wet chemical analysis of samples could be largely deleted. Such a development would make intensive (90% accuracy) overburden sampling more feasible.

OVERBURDEN SAMPLING GEOMETRIC CONSIDERATIONS

It is common practice to divide a mine project area up into a square grid system on a base map so that lines of division are parallel to the existing township-range legal description system. Boreholes for sampling the coal seam and overburden physicochemical characteristics generally fall

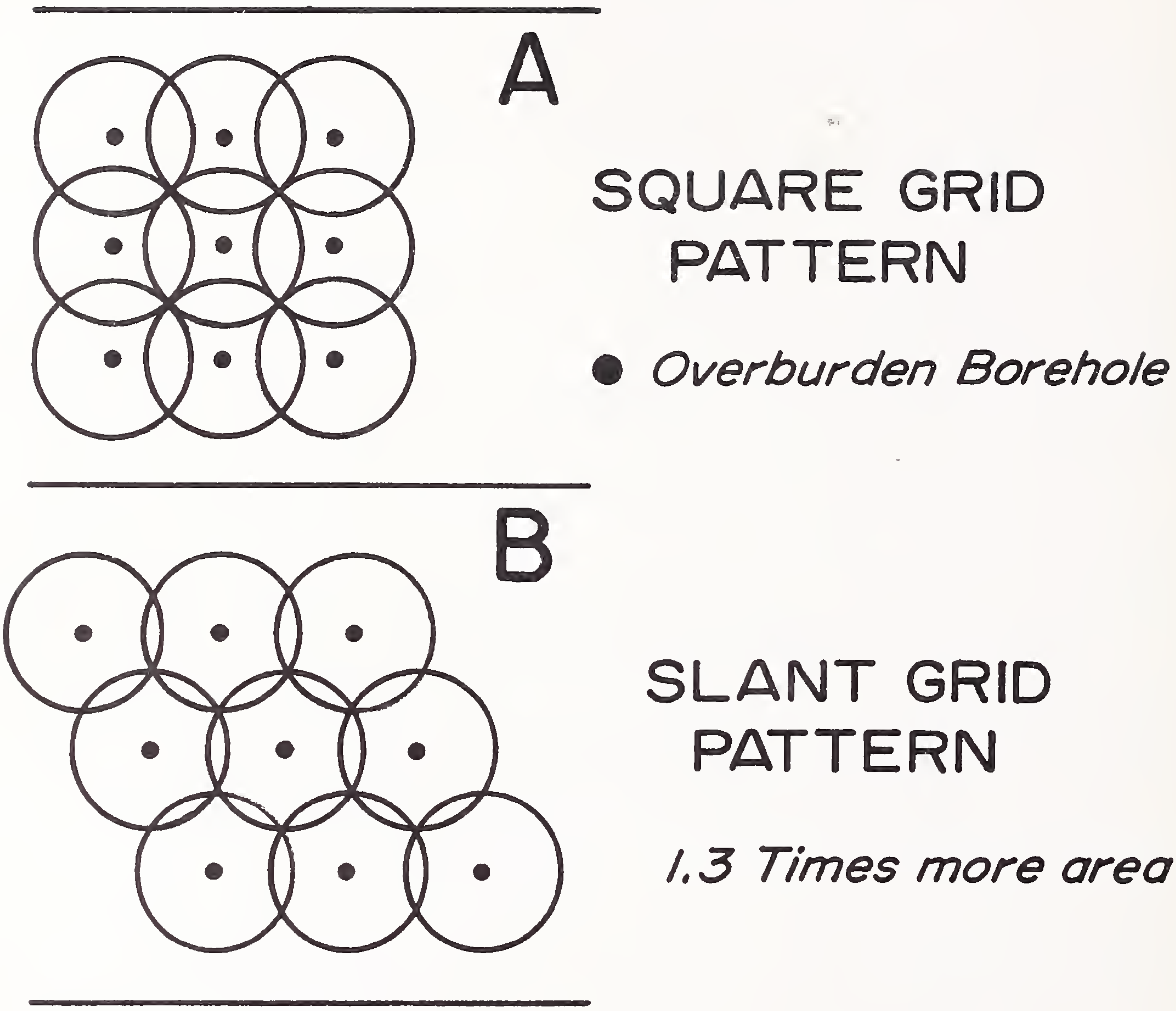


Figure 23. Example of the square grid and slant grid drilling pattern for overburden characterization.

into this square grid system. Since overburden must be sampled at least every 600 m in Montana (MDSL, 1977), a square grid pattern is generally produced with boreholes 600 m apart (Figure 23 A).

When a 600 m square grid is implemented in a project area, it infers that physicochemical results from each borehole must be extrapolated concentrically outward 425 m (radius). This result is shown in Figure 23 A where the circular zone of data extrapolation around each borehole (850m diameter) must exceed the borehole spacing (600 m). In this manner, these 850 m diameter circles provide complete coverage of the project area.

If we assume each borehole in the square grid pattern is representative of overburden material within a radial distance of 425 m, then these same boreholes could be arranged into a 60^{o} slant grid pattern (Figure 23 B), which would cover 1.3 times more project area than the square grid pattern (Schwaller, 1981). The boreholes are spaced 735 m apart in the slant grid pattern, thereby providing greater project area coverage than with a 600 m spacing, yet all portions of the project area are covered with the same 425 m radial distance for extrapolation of data from a central borehole.

When a project area is drilled intensively to attain a characterization accuracy of 80 to 90%, i.e. a 60 m grid, a slant grid pattern could be implemented with boreholes 74 m apart, resulting in efficient overburden characterization with a 30% savings in drilling and laboratory costs.

VI

SELECTIVE BURIAL OF UNSUITABLE OVERBURDEN MATERIAL

There is concern that mining operations may adversely affect the quality of ground-water resources. Part of this concern stems from the belief that ground water will take on, to a certain extent, the chemical characteristics of its flow medium. Additional concern stems from mine spoil research (Arnold et al., 1977; Dollhopf, et al., 1977a) which indicates some spoil profiles lost 10-20 cm of water through the deep leaching process during the hydrologic year, a process that could transport soluble salts and metals downward to the ground-water resource. Through selective dragline handling techniques the demonstration described herein deals with these specific concerns. The objective of this demonstration was to identify a saline overburden zone, bury it in a spoil pile location which will not be intercepted by aquifer reestablishment or plant root development, and then construct a relatively impermeable cap to divert future leaching water around the unsuitable zones.

NORMAL MINE PLAN

The Area B mine site includes approximately 2000 ha of land proposed for eventual strip mining. The research area is confined to the initial 324 ha of Area B.

The Montana Department of State Lands granted a permit to Western Energy Company to begin surface mining operations in Area B in August, 1976. The overburden and exposed coal was to be removed in strips, or passes, that were 45-60 m wide and 1500 m long (Figure 24). Overburden thickness varied from 10 to 45 m in the project area. The first pass (Pass 0) was developed paralleling the 30 meter wide buffer zone along State Highway 39 with further cuts developed parallel to Pass 0 in a southwesterly direction. The same basic mine plan was to be followed through Area B. The overall mining activity, therefore, proceeds in a southwesterly direction away from the highway. Mining operations commenced with the removal of topsoil along and to the east and west of Pass 0. The topsoil was stripped with scrapers and stockpiled in a predesignated area outside of the active mine zone. The depth of the topsoil removed was governed by the soil survey map and averaged about 1.0 m, but ranged from 0.0 m to 1.5 m. After the topsoil was stripped, the overburden was blasted and then removed using a Marion 8050 walking dragline. Following exposure of the Rosebud seam, the coal was loaded with a shovel into 120 ton coal haulers and transported to the coal processing and loadout facilities. As the mining progressed, areas from which the coal was extracted were refilled with spoils, graded to less than a 5:1 contour, recovered with topsoil, and reseeded. The planned post-mine use was grazing.

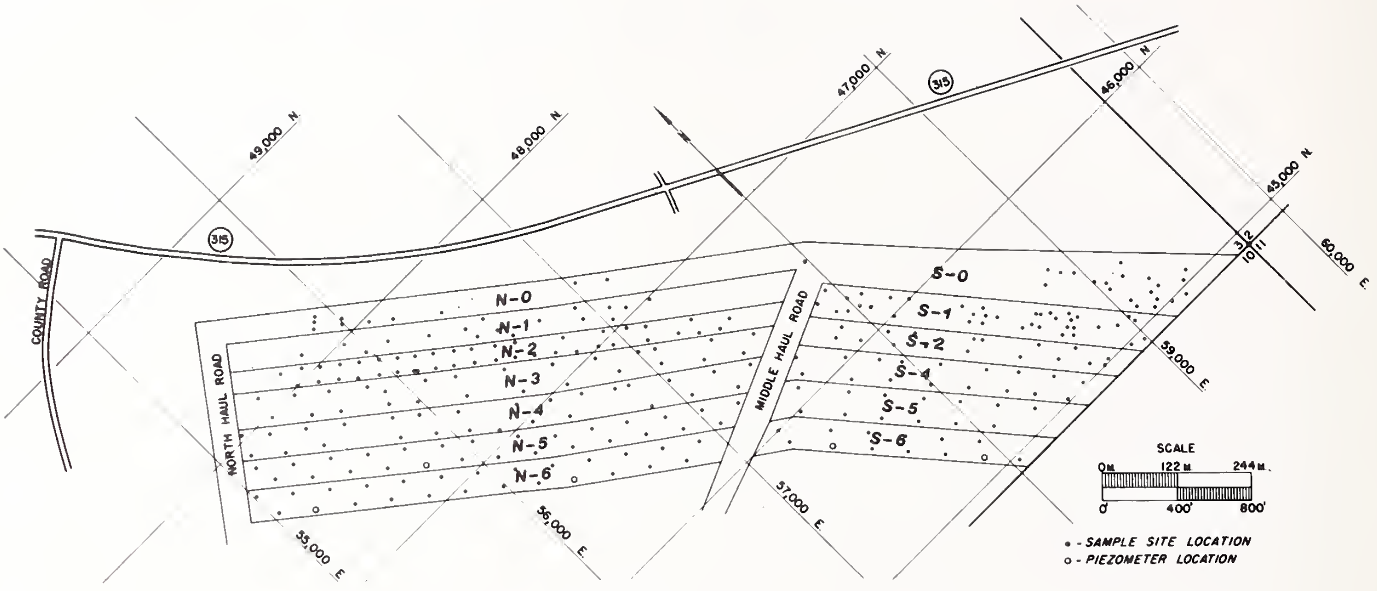


Figure 24. Overview of all mine passes showing sampling site locations in MAES research area at the Western Energy Company Mine Area B near Colstrip, Montana.

DETECTION OF UNSUITABLE OVERBURDEN

As discussed in Chapter V, the project area overburden was drilled on a 305 m (1000 ft) grid and samples were analyzed for 24 different physicochemical parameters. Based on this initial reconaissance it was determined that the parameters EC, Ni, Cu, Zn and clay content often exceeded Montana Department of State Lands suspect levels for overburden materials. In order to implement selective handling of unsuitable overburden materials, the project area overburden was characterized more intensively (60 m grid) and samples were analyzed for only EC, Ni, Cu, Zn and particle size distribution (Appendix C, lab methods).

A sampling technique was devised whereby overburden samples were collected utilizing the Bucyrus-Erie 60-R (BE60-R) drill rig as it prepared the overburden blast holes (Figure 25). The BE60-R was a track-mounted electrically powered drill rig capable of boring a clean hole 33.0 cm in diameter to a depth of 47.6 m. The overburden cuttings from the hole were forced to surface around the drill stem under air pressure of 6 to 30 psi depending on the type of drill bit used. In order to determine the depth at which samples were collected, the mast was marked off in 1.5 m increments using colored weather-resistant tape. As the slide moved down the mast, the location of the drill bit and, therefore, the sampling depth could be accurately determined. Overburden samples were collected from 270 boreholes shown in Figure 24.

A surface deposit of saline overburden was located throughout the mine project area (Figure 26). A 200 m section of mine pass S-4 was chosen to test selective burial procedures (Figures 27 and 28). Because the saline zone tended to dip downward in the overburden immediately south of the middle haul road (Figure 27) it was decided to begin the demonstration approximately 52 m from the ramp road (Figure 29).

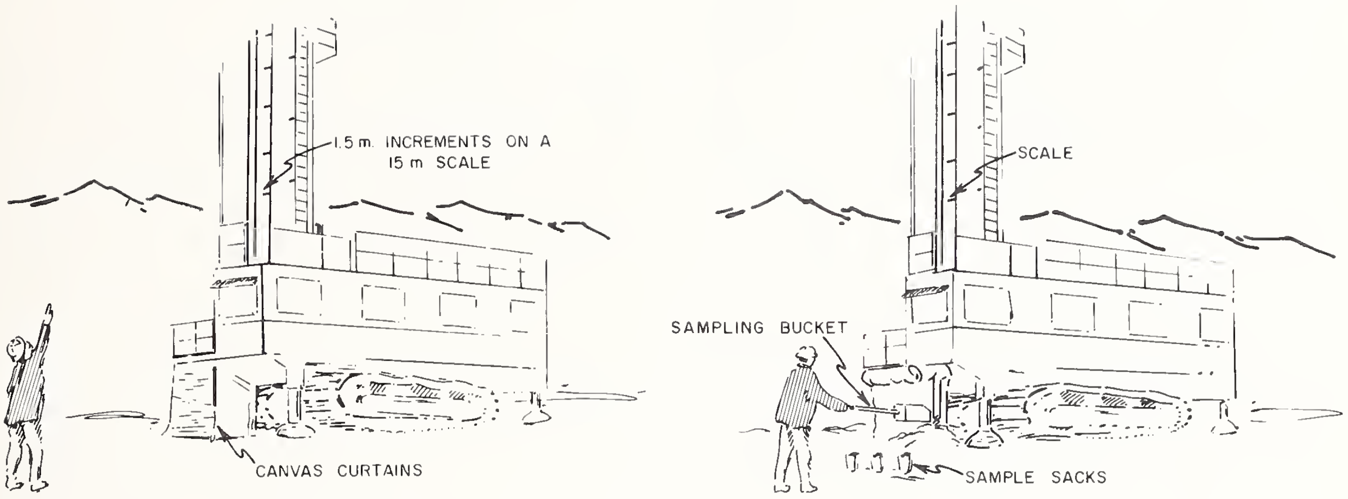


Figure 25. Diagram of the BE60-R drill rig utilized to collect overburden samples.

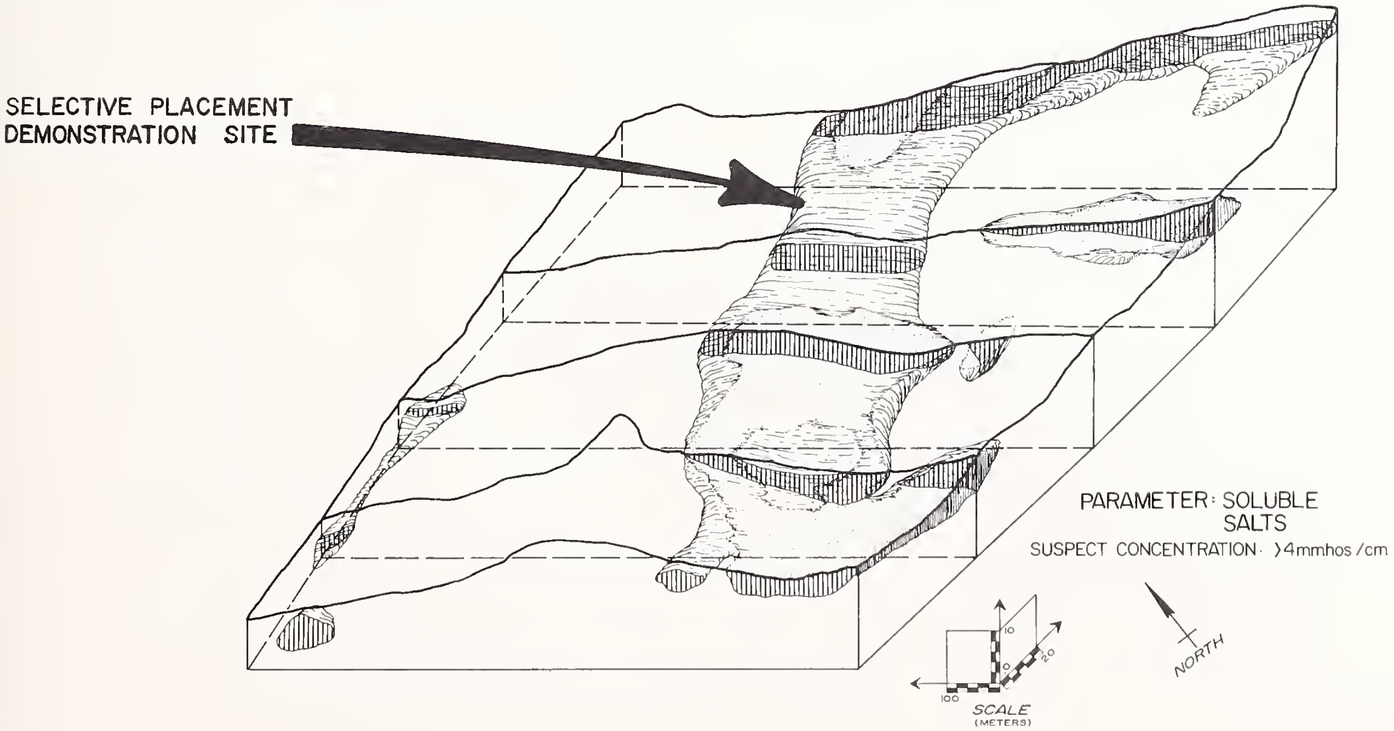

SELECTIVE PLACEMENT DEMONSTRATION SITE
PARAMETER: SOLUBLE SALTS
SUSPECT CONCENTRATION· >4mmhos/cm
NORTH
10
20
0
100
SCALE (METERS)

Figure 26. Overburden fence diagram showing location of saline material in eight mine passes (see Figure 16) at the Western Energy Company Mine Area B, Colstrip, MT.

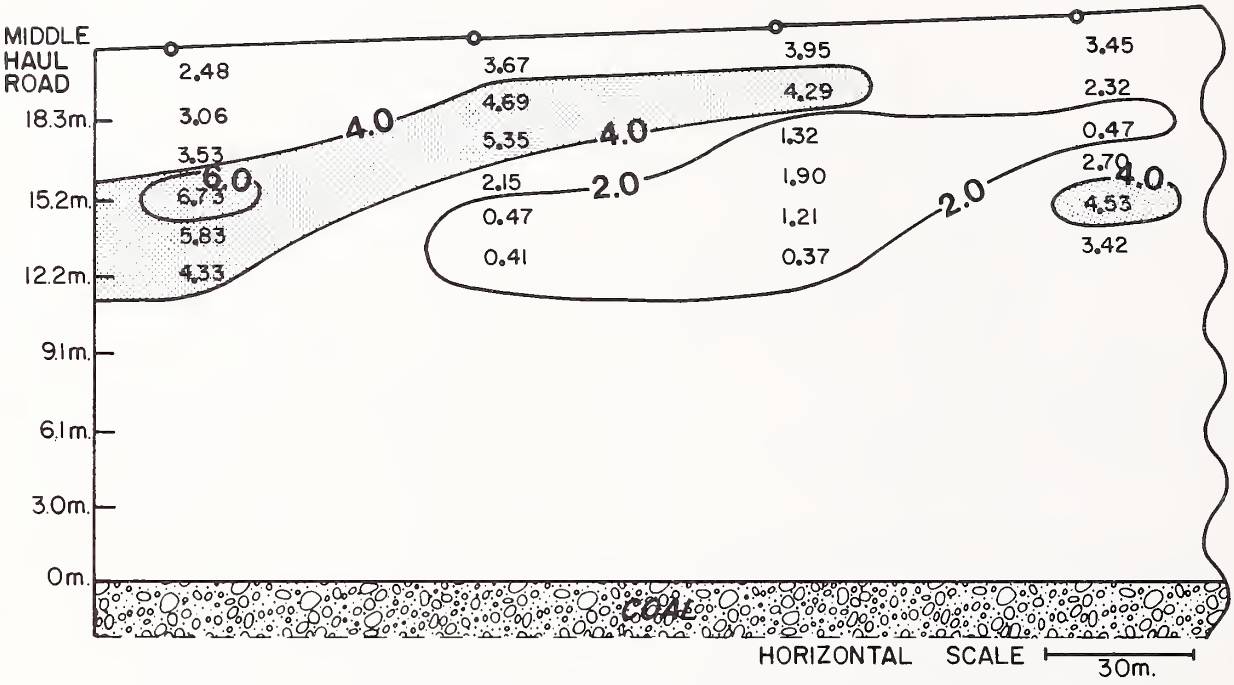


Figure 27. Overburden distribution for electrical conductivity in the east half of pass S-4. Isoline interval is is 2 mmhos/cm and the suspect concentration is 4 mmhos/cm.

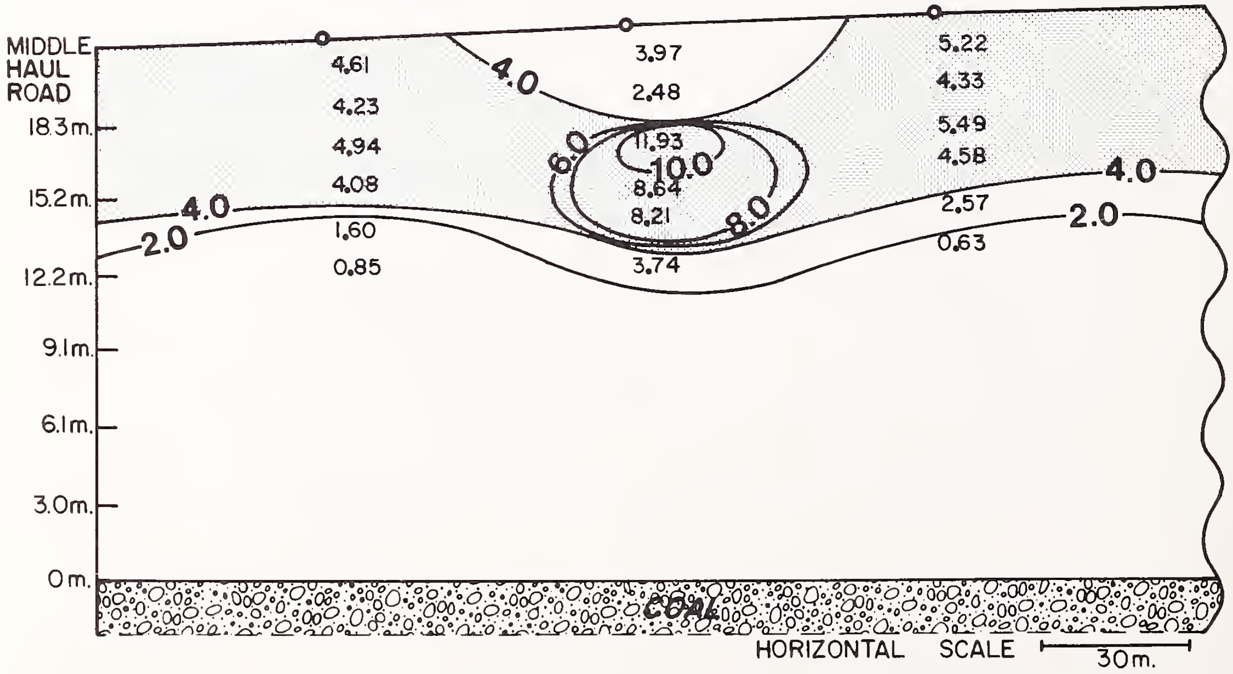


Figure 28. Overburden distribution for electrical conductivity in the west half of pass S-4. Isoline interval is 2 mmhos/cm and the suspect concentration is 4 mmhos/cm.

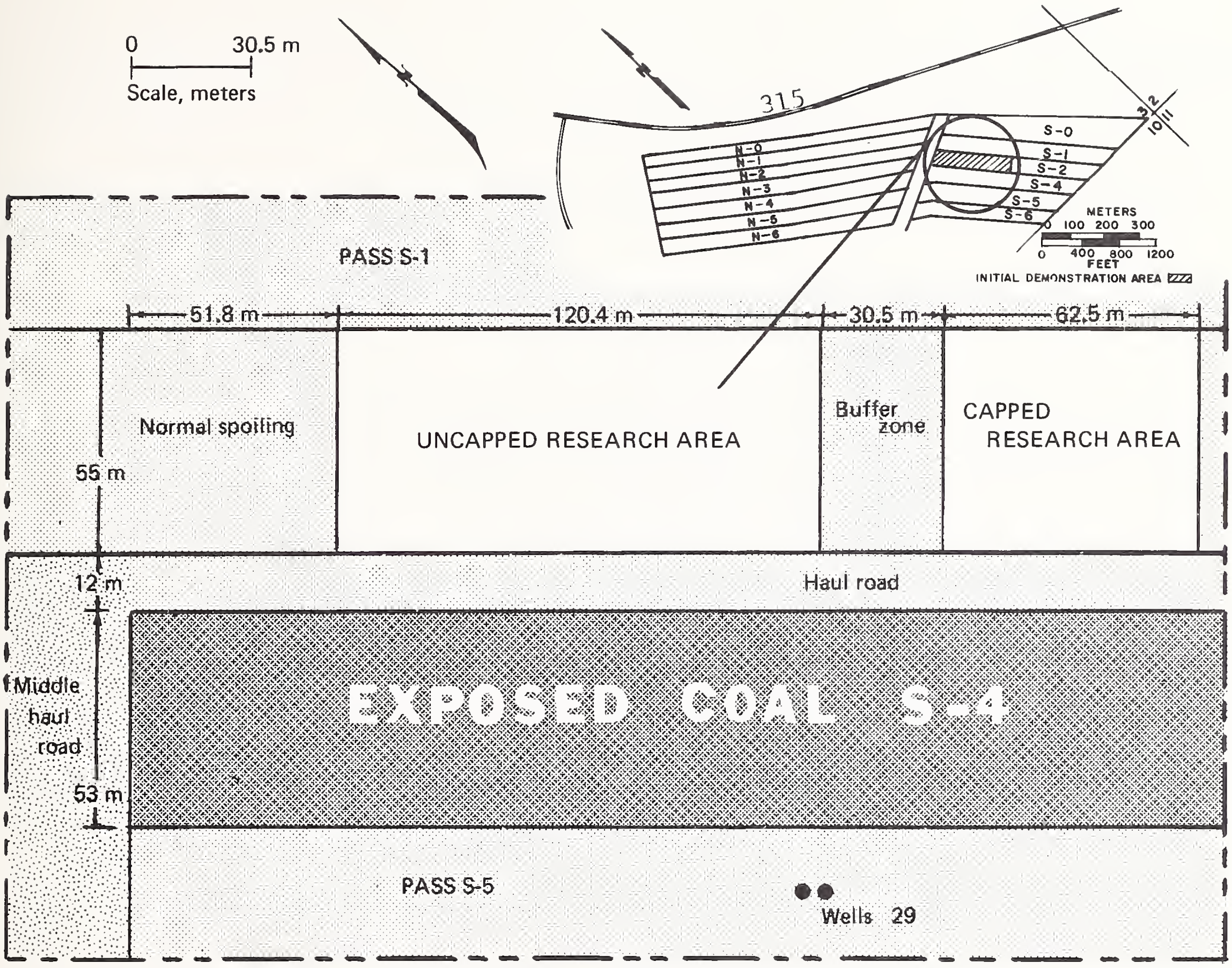


Figure 29. Orientation and dimensions of selective handling demonstration. Pass S-2 void received the spoils from Pass S-4.

SELECTIVE OVERBURDEN HANDLING PLAN

The selective handling and segregation of the saline material of Pass S-4 required several distinct steps. These steps included:

1) identification, selective stripping and stockpiling of the surface saline material;
2) construction of a foundation base in the S-2 pit bottom with nonsaline overburden;
3) placement of the stockpiled saline material on the foundation;
4) shaping the deposited saline material prior to receiving the clay cap;
5) construction and compaction of the clay cap; and
6) burial of the capped saline material using the balance of acceptable overburden.

It should be noted that in the following discussion two burial type selective handling demonstrations are discussed: One demonstration with a clay cap over saline material and one without the clay cap (Figure 29). It was anticipated that the addition of the uncapped study to the total demonstration would provide a means to: 1) physically work out the details of constructing the capped study without the difficulty of actually building the clay cap, 2) evaluate the effectiveness of simple burial of saline material especially with regard to its eventual effect on ground water, and 3) allow a comparison between a capped and uncapped study to assess the effectiveness of hydrologically isolating the saline material.

SELECTIVE HANDLING PROCESS

Overburden in the first 52 m south of the middle haul road in Pass S-4 was stripped and spoiled in the normal manner (Figure 29). At the 52 m mark, the operation was changed to accommodate the intended selective handling process. The dragline walked south 10 steps (about 1.8 m per step) on May 16, 1977 and began stripping the surface saline material to a depth of approximately 4.5 m. This material was deposited on the highwall of Pass S-5 50 m to the rear (south) of the dragline (Figure 30). The height of the dragline bucket was about 3.0 m. Therefore, to approximate the 4.5 m stripping depth, 1.5 times the bucket height was the common guide.

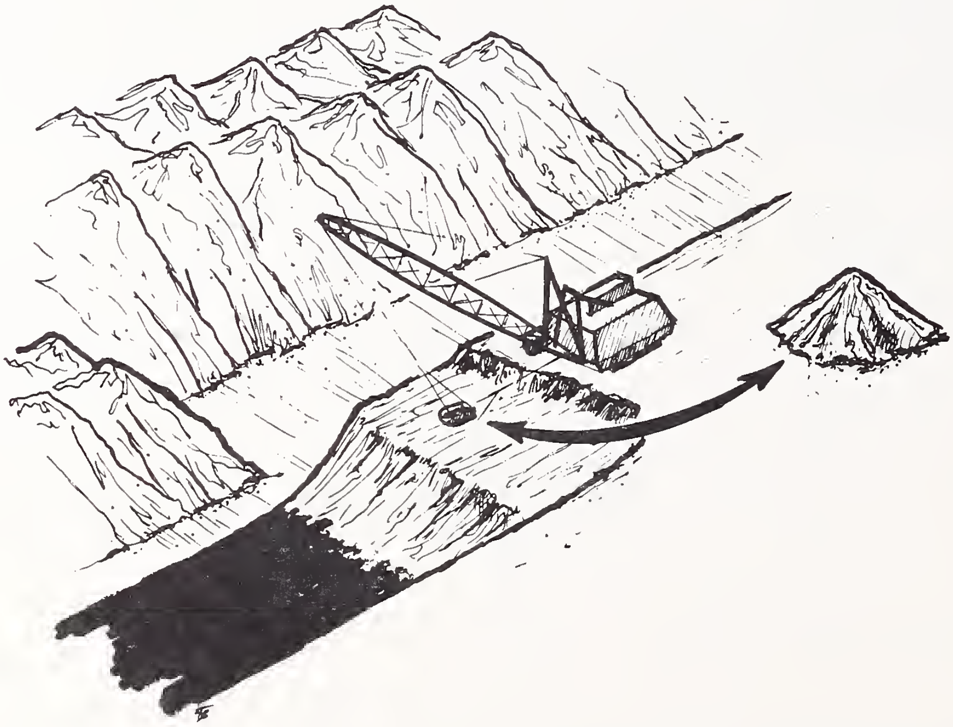


Figure 30. Initially the dragline stockpiled the surface 4.5 m of salt material on the S-5 highwall.

After all the saline material that could be reached had been stripped and deposited on the highwall, the dragline moved forward (north) 2 steps.

The net result of the 10 step move to the south and the 2 step move to the north was 8 steps south. This constituted a total move about 15 m. The usual move under normal stripping operations is between 13 and 18 steps depending on the overburden depth. According to the mining engineer, the shortened move was necessary to insure proper placement of the materials in the Pass S-2 pit. The deviation from the norm was 9-18 m per move.

After the 2 step move to the north, the dragline stripped the remaining nonsaline overburden and placed it in the pit bottom as basement material (Figure 31). Flags were placed along the S-2 spoil ridge 12 m above the pit bottom to guide the dragline operator in placing the basement material. Hydrologic investigations prior to the demonstration indicated chemically unacceptable overburden should be elevated at least 12 m above the pit base to prevent contact with the aquifer that might reestablish.

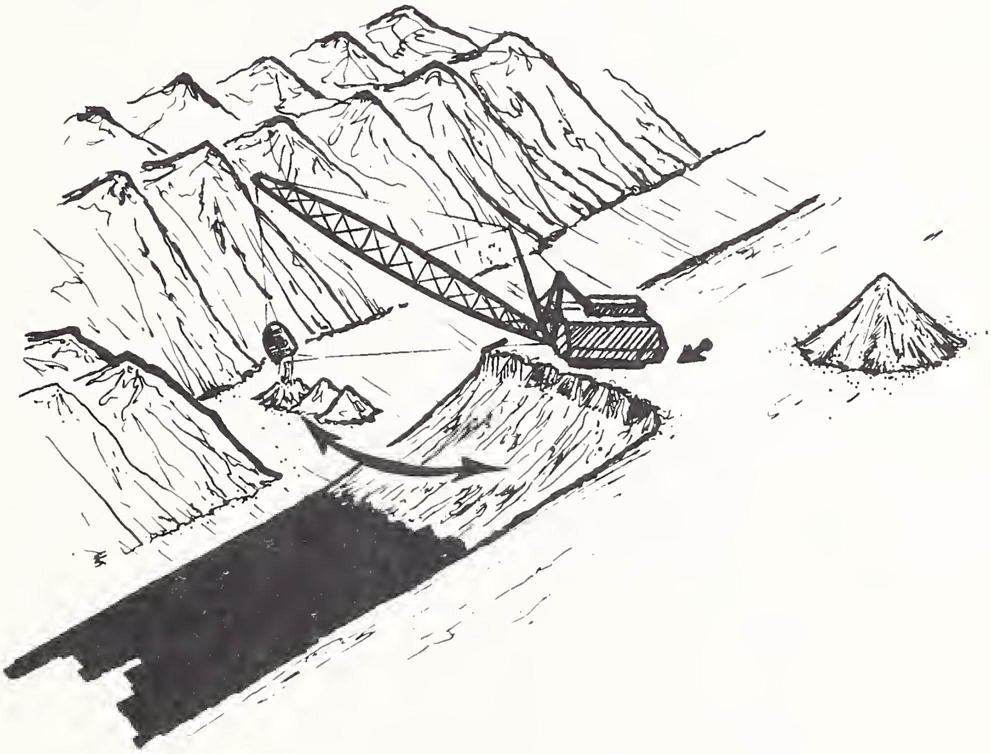


Figure 31. Overburden below the surface salt affected zone was placed in the S-2 pit bottom as basement fill.

Due to relatively heavy precipitation received on May 15 (4.01 cm) and during the entire demonstration (a total of 6.48 cm), a large quantity of water accumulated in the pit. Continuous pumping removed part of this water and the remainder was largely displaced to the south as the basement was constructed. Throughout the length of the demonstration, great care was taken to be sure none of the water was trapped under the spoils, since this could have the effect of both decreasing the spoil pile stability and confounding the ground-water monitoring program.

As work continued on construction of the basement, the dragline operator would occasionally use the bucket to level and shape this basement material (Figure 32). The dragline bucket was used to create a slight depression in the center of the basement material to receive the saline

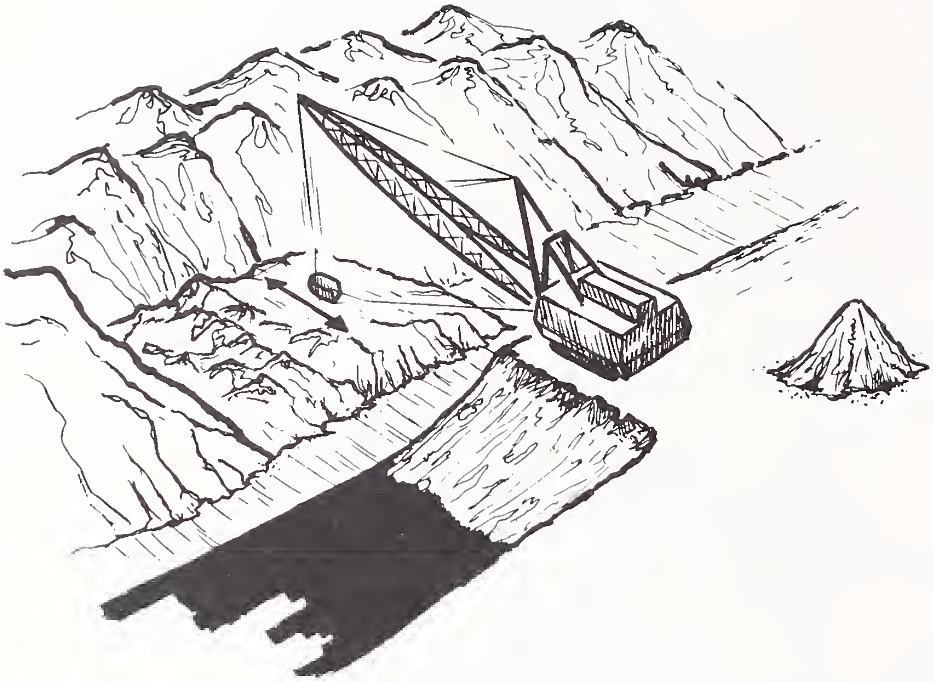


Figure 32. Levelling and shaping the basement material with the dragline to reduce the work required by a dozer in grading these materials.

overburden. The time required to complete stripping the saline material and the nonsaline overburden during the first move was approximately 5.5 hours.

A second 10 step move to the south was made and stripping of the surface 4.5 m began. Rather than place this saline material on the S-5 highwall, as was done previously (Figure 30), it was deposited directly on the prepared basement area (Figure 33). By going directly from the

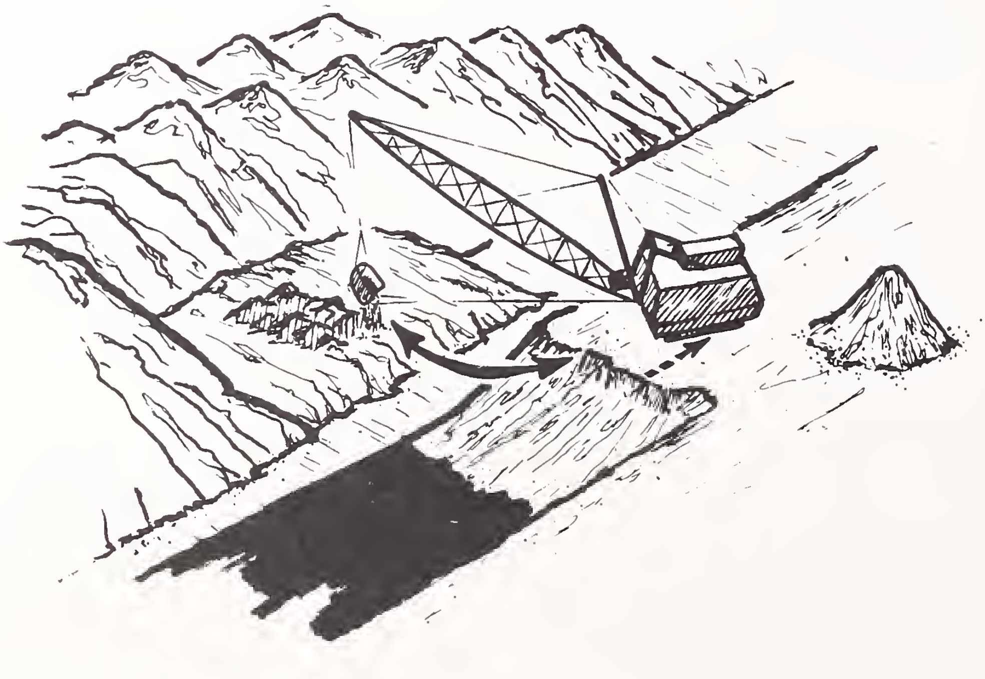


Figure 33. Direct deposition of saline material on the basement fill.

S-4 overburden to the S-2 pit, considerable material re-handling was eliminated and the total operational costs reduced. The objective was to build the saline material to a thickness of about 6.0 m above the basement material and then work the material to a 5:1 grade using a D-9 dozer (Figure 34). The intent was to envelope the saline material with clean material as shown in Figure 35.

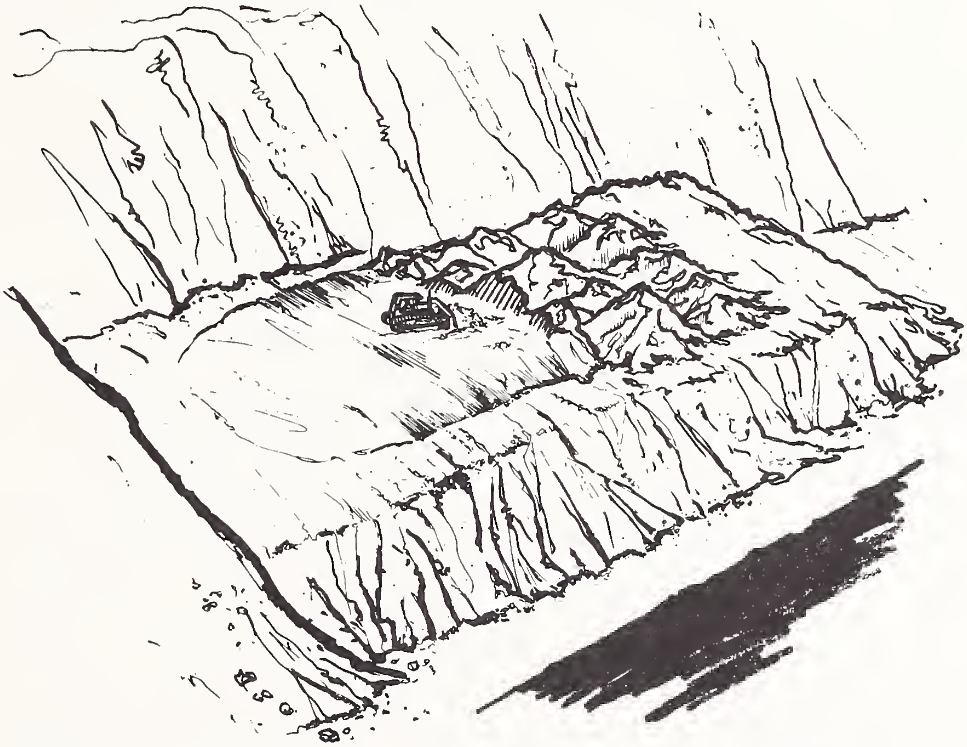


Figure 34. The salt affected material was shaped to a 5:1 grade with a D-9 dozer.

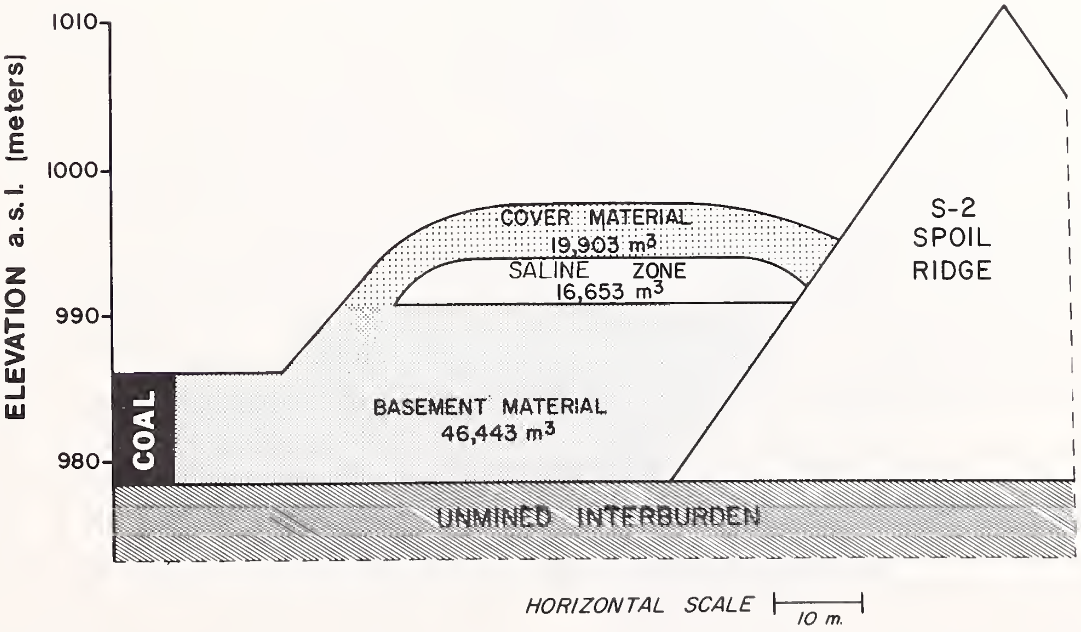


Figure 35. Transverse cross-section of uncapped demonstration area.

As the zone available for deposition of saline material was filled, the dragline extended the basement with nonsaline overburden. As soon as sufficient foundation material was placed in the pit bottom and leveled off, more saline material was stripped and deposited on the bench. This process of alternating from non-saline basement material to saline material and back, was continued until all of the saline surface of Pass S-4 was removed. The dragline moved north 3 steps and the overburden material was used to extend the basement.

The dozer assigned to work the spoiled material (pit dozer) cut an access road between the spoil ridges to enter the demonstration area. After the dragline moved 13 steps south the last of the saline zone was placed on the basement fill and the dozer began working this material to the proper grade. The final depth in the center was about 4.5 m and sloped down to the east and west.

While the dozer was working in the pit area, the dragline continued to lay down basement material. Because of the dozer's position under the dragline boom, the speed of the dragline operation was greatly reduced. The dragline operator had to modify his normal boom swing to avoid swinging the bucket over the dozer. To avoid an unsafe situation, the bucket was filled and then kept close to the dragline body as the boom was swung over the pit area. Once past the vicinity of the dozer and personnel working below, the bucket was allowed to swing out and then the material was dumped. When there was not a safety conflict with the nurse cat, the dragline boom was rotated in the opposite direction over the highwall instead of the pit. "Nurse cat" refers to the D-9 dozer which assists the dragline in highwall trimming and pad preparation.

When the dozer had completed contouring the saline material, the dragline buried the material with nonsaline overburden (Figure 36).

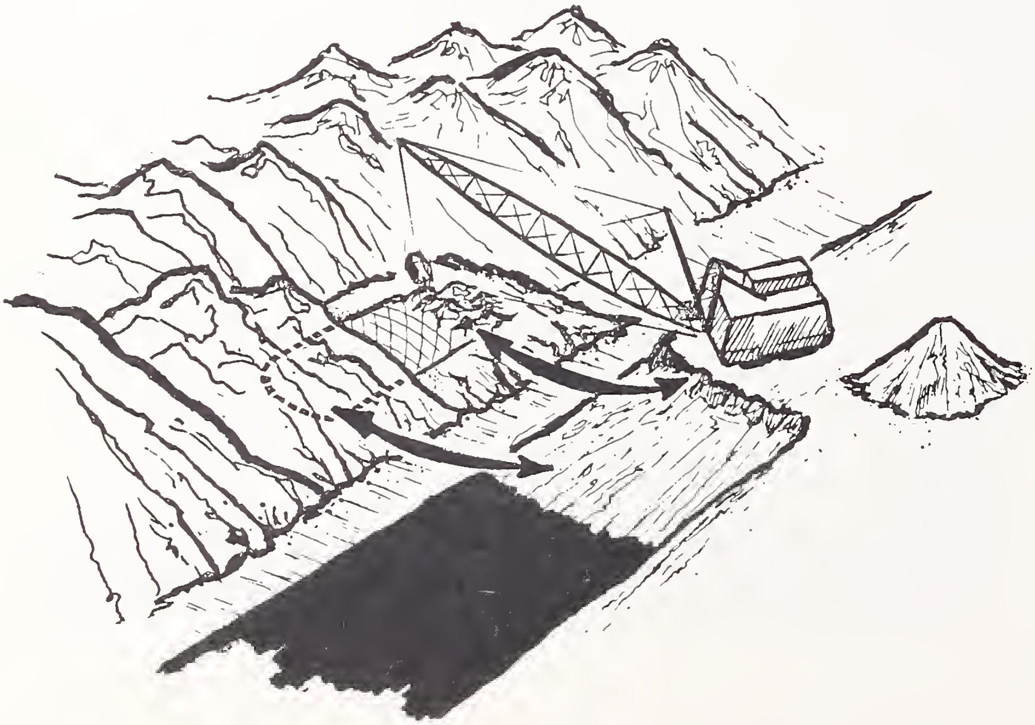


Figure 36. Burial of saline material (lower arrow) with nonsaline overburden.

The dozer moved to the south end of the basement bench and leveled that material. Cover material was deposited by the dragline to a depth of 4.5 to 6.0 m over the saline material starting at the extreme north end and working south. Once sufficient cover was laid down, the dragline continued the basement extension while the dozer worked the cover to a 5:1 grade. The balance of the overburden was used as basement material. The time required to strip and selectively place the overburden material on this move was about 10.0 hours.

The dragline moved 13 steps south. The saline surface material was stripped and deposited tightly against that already on the basement. After the saline material had been stripped, the dragline moved north 3 steps resulting in a net 10 step move rather than the previous 8 step move. The dragline operator indicated that the longer move, although still less than normal, would probably work somewhat better than the shorter move. The mining engineer agreed that the 8 step move was probably a little too short.

Construction of the basement bench was resumed. Because of the dozer's position in the pit beneath the boom, it was not possible to cast the overburden material to the far eastern side of the pit. The dozer had to be moved off of the deposited saline material to allow the dragline to swing and "cast" the basement material toward the bottom of the S-2 spoil ridge. Once sufficient basement material had been cast to the far side of the bench, the material was worked to the proper configuration by the pit dozer.

Sufficient basement material had been laid down at this point to accomodate more saline material. Most of the saline material originally stockpiled on the S-5 highwall was rehandled and deposited tightly against the salty spoil already in position (Figure 37), and the pit

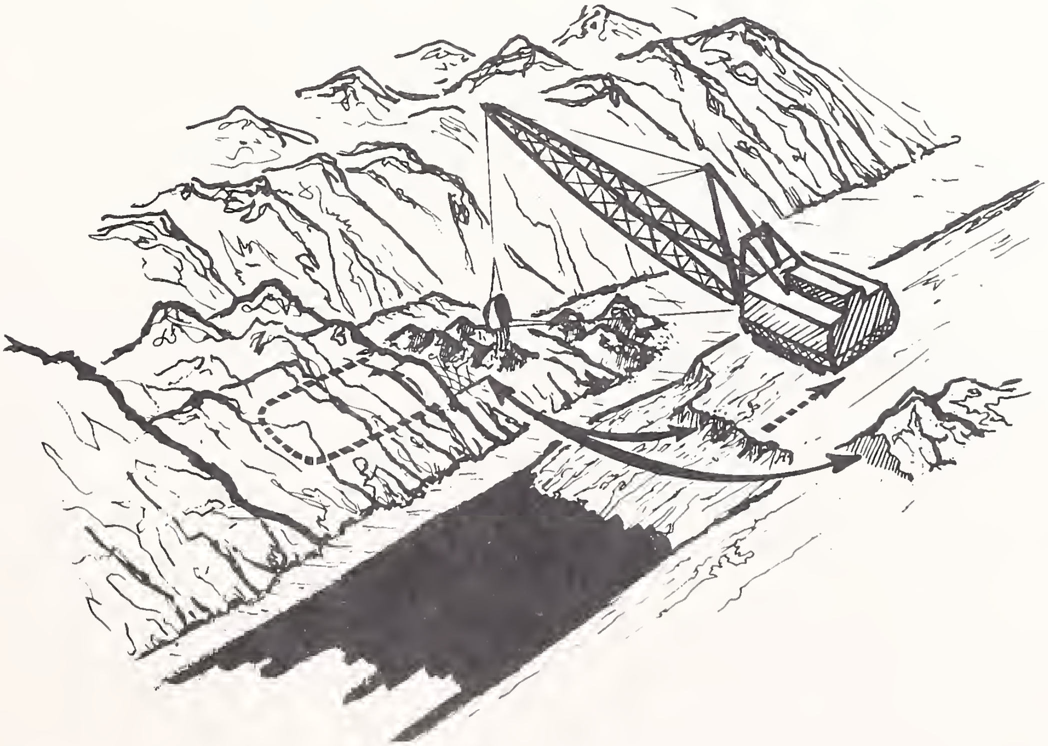


Figure 37. Deposition of saline material from both the overburden and stockpile.

dozer worked it to the required grade. While the dozer worked in the pit, the dragline deposited more basement bench. After the saline material was adequately graded, the dragline buried the zone with nonsaline overburden. The balance of the overburden material accessible to the dragline on this move was used to cover the saline material in the pit. The time spent actually stripping overburden on this move was about 8.5 hours.

The dragline walked south 13 steps and began stripping saline material. This material was deposited on the basement fill in what was the last part of the uncapped study area (Figure 38). The dragline bucket was used to spread the deposited material as much as possible before the pit dozer was sent in. This use of the bucket, which had also been done previously, was intended to reduce the work required of the pit dozer in grading the materials. Not all of the surface saline material could be placed on the bench designated for the uncapped study area. Therefore, the remainder of the saline material was spoiled on the S-5 highwall behind the dragline body, for later rehandling (Figure 39).

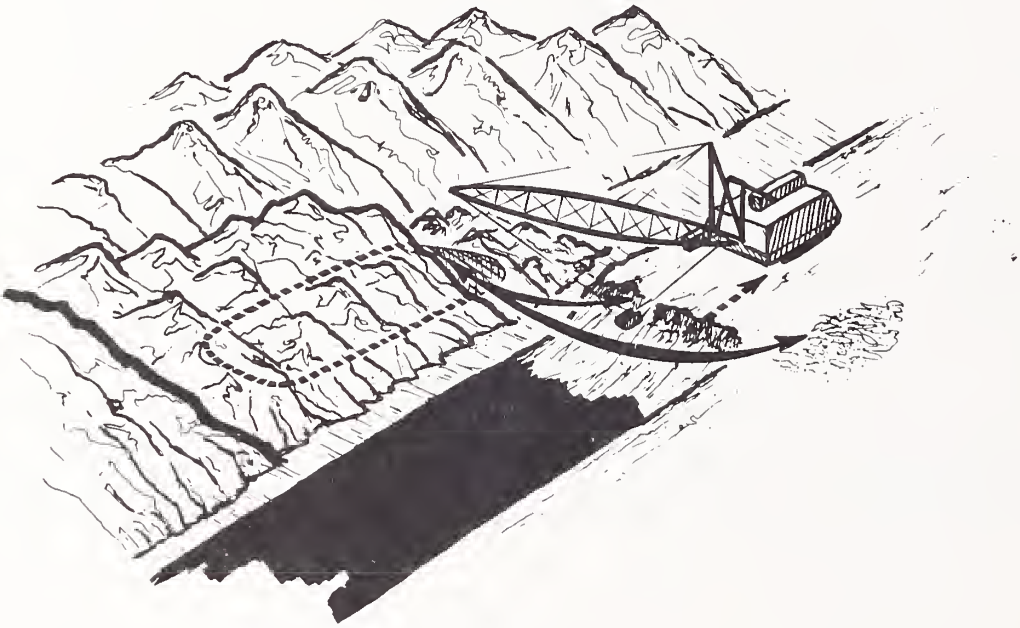


Figure 38. Most of the highwall stockpile was deposited on the basement fill towards the completion of the uncapped study.

A measurement of the study area constructed thus far indicated it was about 120 m in total length (Figure 29). This was larger than originally intended by about 45 m. When viewed from the dragline cab, the area appeared much smaller than the measurements indicated. Apparently this distortion contributed to the over-construction of the area. However, it was not felt that the increased size would create any special problems.

It was also evident at this point that deposition of the spoiled material had been stretched too far down the pit (to the south). The situation was referred to as "being long on dirt" in the S-4 highwall and, if not corrected, could lead to the dragline becoming "dirt bound". This implied that the dragline would have nowhere to place overburden material within its boom radius. In order to avoid a "dirt-bound" sit-

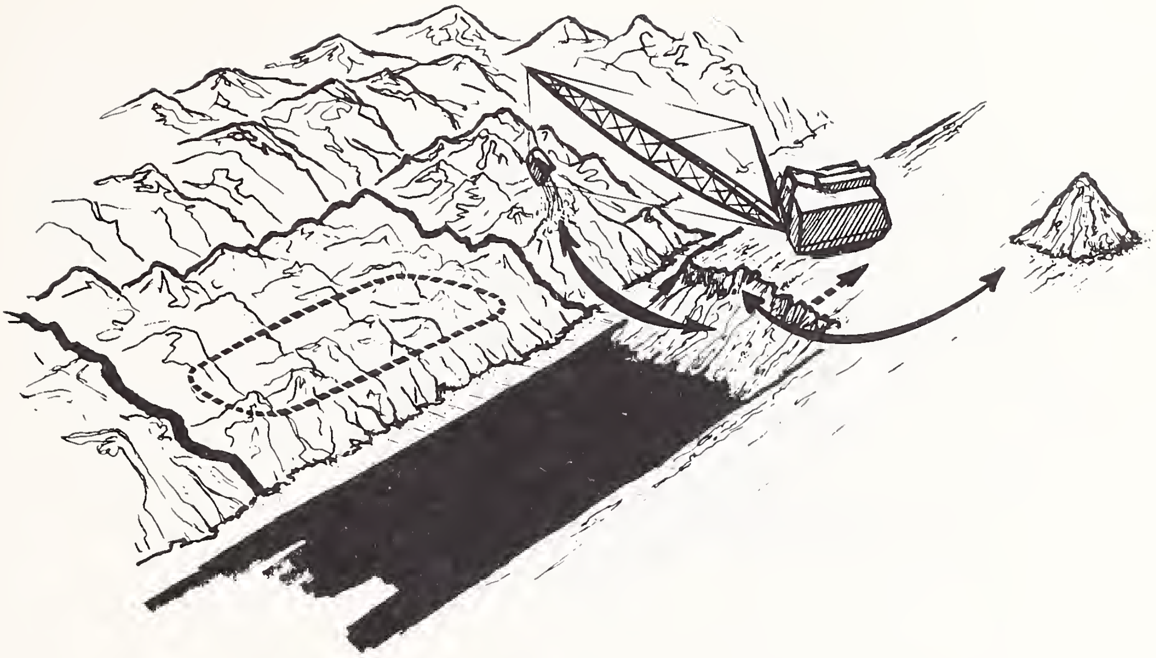


Figure 39. A buffer zone was constructed when the uncapped study was completed. During this phase the surface saline zone was stockpiled on the highwall.

ation, which would lead to considerable and expensive downtime, it was necessary to modify the operation.

With the completion of the uncapped study area, the next phase of the demonstration called for the construction of a buffer zone to separate the uncapped and capped areas (Figure 29). The buffer zone was to be built of nonsaline overburden material and the elevation (above sea level) was to coincide with that of the surface of the uncapped area. To dispose of as much of the undisturbed overburden as possible, and thereby correct the possible dirt-bound condition, it was decided to increase the elevation of the buffer zone.

The balance of the overburden was stripped off the coal and deposited in the buffer zone. The work time to complete overburden removal in this move was about 10.5 hours. The dragline moved 13 steps south and began stripping the surface 4.5 m of overburden. The saline material was deposited in the second highwall stockpile. After the saline material was removed, the nonsaline overburden below was used to complete the cover over the uncapped area and to continue construction of the buffer. The time required to complete stripping this overburden was about 8.5 hours.

The dragline walked to the south 12 steps and the saline material was stripped and added to the second highwall stockpile. Deposition of the nonsaline overburden in the pit bottom commenced after the dragline moved north 2 steps. The basement fill for the capped study was started using this nonsaline material (Figure 40). As the dragline was still a little "long on dirt" in the highwall, the flags indicating the depth of the basement were raised 3.0 m. This adjustment provided a foundation of about 15 m rather than the 12 m constructed under the uncapped area. Increasing the basement depth allowed more of the overburden material to be spoiled within the working radius of the boom. It was felt that the difference in basement thickness between the two demonstrations

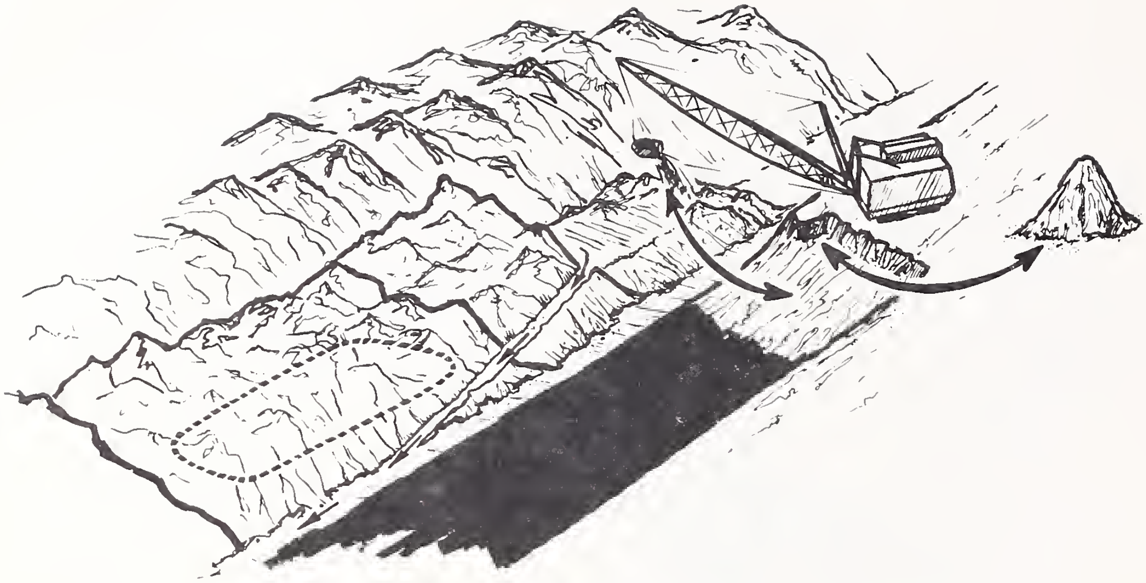


Figure 40. Following construction of the buffer zone, a 15 m thick basement fill for the capped research area was deposited in the pit bottom. During this process the surficial saline zone was stockpiled on the highwall.

would not introduce significant experimental error in the scientific comparison of the two techniques.

During the initial construction phase of the capped study area, work began on the scraper access road from the clay deposit in Pass N-3 to the demonstration area. A team of three scrapers and two dozers were used to construct the road in such a way that traffic to and from the demonstration area would not conflict with the coal haulers working in Pass N-3. Scoria (clinker) road material was laid down on the part of the road leading directly into the uncapped area. Elapsed time spent in actual overburden removal with this move was approximately 10.0 hours.

The dragline walked south 13 steps and the bucket was used to level the top of the basement material in the pit. This was done to assist the pit dozer which had to shape the basement fill to receive the first part of saline material in the capped study. Because of the position of the working pit dozer, the dragline had to shut down. The shutdown was the first time during the demonstration that the dragline could not work at all and lasted 1.6 hours. To expedite the leveling of the basement bench, and reduce the dragline downtime, the nurse cat was instructed to assist the pit dozer.

When the basement bench was completed for the capped study, the dragline stripped the saline surface material and deposited it on the basement fill (Figure 41). Deposition of the saline material started about 4.5 m south of the point of contact between the buffer zone and basement bench. The reason for this indentation was to allow the saline material to be enveloped between the basement and the clay cap cover materials, similar to what was done previously in the uncapped study.

From this point forward, no further selective stripping of the saline zone from the overburden was done. Results of the salt analyses from the first 200 m of Pass S-4 had indicated that the surface 4.5 m

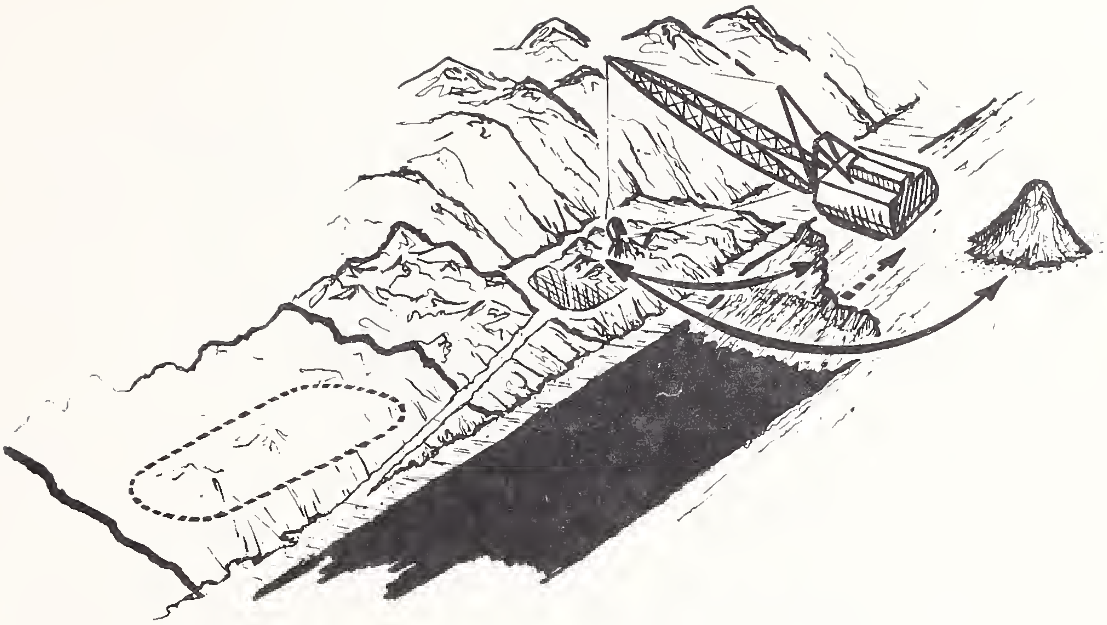


Figure 41. When a portion of the basement bench was completed for the capped study, the dragline deposited saline material over the basement from both the overburden and stockpile.

saline zone tended to taper off at this point to only the surface 1.5 m of material. The rest of the saline material for the capped study was taken from the stockpile on the Pass S-5 highwall.

Now that the saline zone stripping was completed, the dragline removed the lower nonsaline material. This material was deposited on the buffer zone and allowed to overlap onto the uncapped area. As the dragline worked this material the pit dozer shaped the saline material prior to the construction of the clay cap. By midmorning of May 19, the saline material was ready to receive the clay cap. The dragline shut down throughout this operation. Since the dragline was down for the duration (5.75 hrs) of the construction of the first phase of the cap, the downtime was utilized to do routine maintenance and an electrical check-out of the dragline.

Construction of the clay cap (Figure 42) was completed in two phases (Figure 43) due to limitations of the dragline reach. In the first phase, the north half of the cap was laid down, compacted, and then buried (Figure 43). The saline material in the southern half portion was deposited on the bench of basement material and in contact with the initial saline material. A dozer was used to expose the clay cap of the first phase by pushing the cover material to the west toward the S-4 highwall (Figure 43). Approximately 5.5 m of the Phase I cap was thus exposed. Scrapers then began laying down the clayey material for the Phase II cap. The clay was laid across the saline material, in a similar manner to that of Phase I, and on top of the 5.5 m of the exposed clay cap edge. This created a well-compacted "overlap" of the two phases (Figure 43). The entire area was then buried with nonsaline cover material to create a single, continuous capped zone.

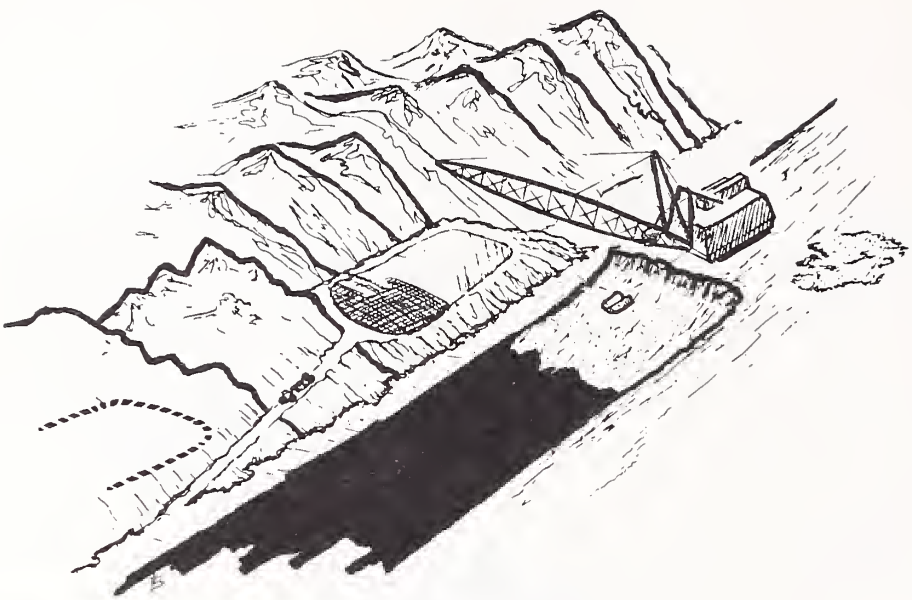


Figure 42. Clay located 1000 m from the demonstration site was applied over the saline material to a depth of 0.7 m by scrapers. The dragline had to shut down during this clay capping operation.

The clay material used for the cap demonstration was located in the northern part of the third pass (N-3) about 1000 m from the clay cap site. This zone, which rested on the coal seam, was about 3 m thick by 18 m by 62 m in dimension. The material had an average clay content of 49.6% by weight, determined by the hydrometer method.

Table 11 summarizes the clay mineral composition of the capping material. The low permeability of clay (0.85 cm/day) made it the most obvious choice for capping material. The major clay minerals found were illite, and kaolinte, with lesser amounts of smectite, chlorite, and quartz. Sample number one was somewhat of an exception since its composition was dominated by smectite (36%), but substantial amounts of illite (30%), and kaolinite (23%) were also present.

Table 11. Estimated clay mineral composition of the capping material.

	Percent					
Sample	Smectite	Illite	Vermiculite	Chlorite	Kaolinite	Quartz
1	36	30	0	3	23	7
2	4	38	0	9	35	14
3	21	37	0	7	22	14
4	4	39	0	10	26	22
5	11	33	0	5	33	19
6	3	35	0	5	46	12

* Analysis performed with X-ray diffraction methodology at Montana State University

+ Classes for Numerical Values

100%-75% Very High
75%-50% High
50%-25% Moderate
25%-5% Low
5%-1% Trace

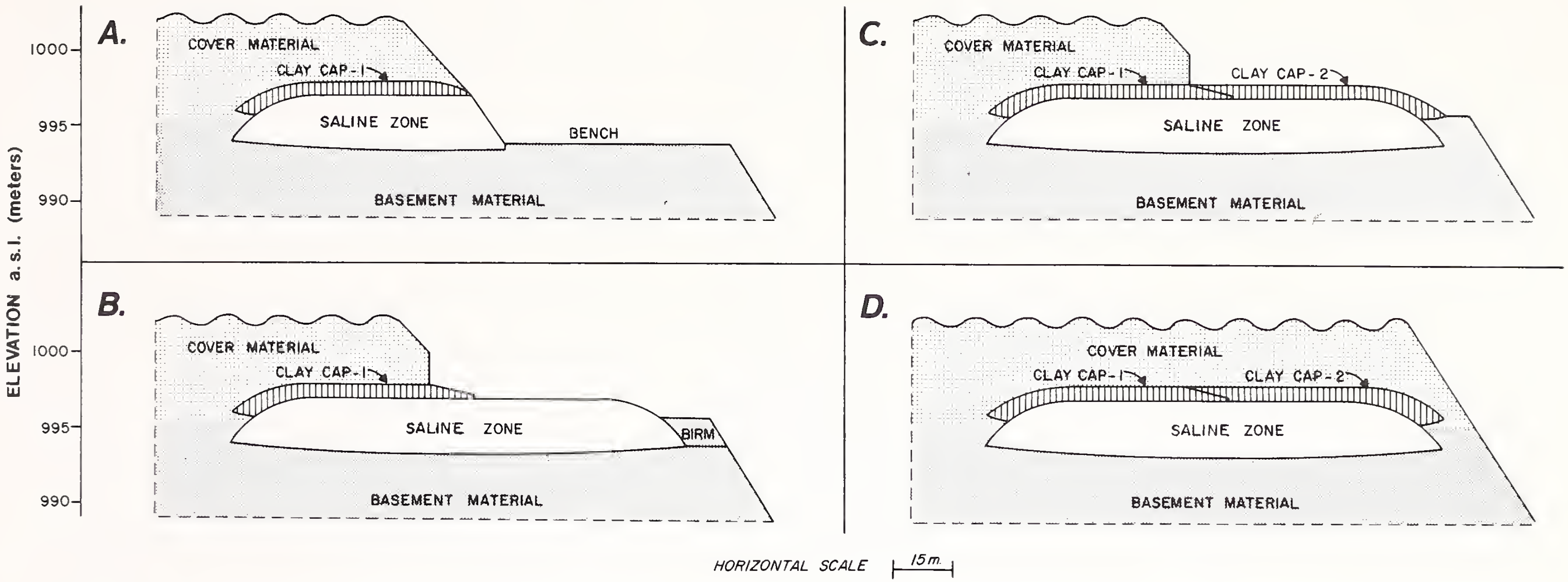


Figure 43. Clay cap construction was completed in two phases. A) first half of cap constructed and covered, B) part of cover material was dozed away to expose clay cap, C) second half of cap locked into first half, and D) entire cap covered with spoils.

The generalization can be made that soils dominated by clay minerals with a 1:1 lattice structure of tetrahedral to octahedral layers of molecular structure (i.e., kaolinite) will have a greater hydraulic conductivity than soils dominated by clays with a 2:1 lattice structure (i.e., smectite), composed of tetrahedral-octahedral-tetrahedral arrangement. The 1:1 lattice structure inherently contains hydrogen bonding between layers which holds successive units together. This bonding has two effects. First, it tends to favor the formation of layer particles, and second, water cannot permeate successive units of the particles to cause expansion and contraction with wetting and drying. This results in a relatively stable soil structure; one which will conduct water more readily than 2:1 lattice clay systems. The 2:1 lattice clays do not have this hydrogen bonding effect and the units readily expand and contract with wetting and drying creating a formation that tends to be impermeable when wet. These 2:1 lattice clay particles also tend to be smaller in dimension than those of kaolinite.

Illite is a 2:1 lattice clay mineral but has characteristics different than those described above. Potassium ions are lodged between adjacent layers and the common attraction of these potassium ions by adjacent units holds them tightly together by forming a chemical bonding mechanism called a "potassium bridge". The lattice, therefore, is relatively nonexpanding.

Therefore, construction of a clay cap with the intent to divert water would best be made with a soil system dominated by a smectite clay. The mechanics of this field demonstration dictated that the earlier described clay deposit, dominated by kaolinite and illite, be used for the clay cap demonstration. Even though the capping material was not of smectite, this capping material was nearly 50% clay, an unusually high percentage, and when properly compacted, it should have a slow hydraulic conductivity.

The moisture-density relationship of the clay cap material (Figure 44) indicated that capping material should be at a moisture content of 14.6% by weight to obtain maximum compaction. The standard Proctor density (Figure 44) was 1.79 g/cm^3 (112 pcf).

On May 7, 1977, the clay deposit was very dry, probably only a few percent water by weight, had a very hard and rocky structure, and obviously would not lend itself to compaction after being transported by scrapers to the demonstration area. Therefore, the ripper blade of a D-9 dozer was used to loosen the clay and a sprinkler system was constructed on the clayey material for irrigation from May 10 through May 12. In addition, precipitation contributed 6.48 cm from May 15 to May 19. Field observation of the clay deposit indicated the increased moisture content changed the clayey material from a rocky substance to a soft and pliable mass. Following the irrigation and natural precipitation, part of the clay deposit was apparently too wet for maximum compaction. The moisture content of the near surface of the clay deposit was about 25% water by volume which is equivalent to about 20% water by weight. A Troxler gauge with radioisotope sources was used to monitor the clay moisture-density characteristics. The moisture-density characteristics of the clay deposit below the 20 cm depth were unknown, initially. The density of the clay deposit from 0-20 cm ranged from 1.19 to 1.64 g/cm^3 and averaged 1.4 g/cm^3.

The first phase of the cap was constructed with 71 scraper loads of clayey material each 16.8 m^3 (22 yd^3) in volume. As discussed pre-

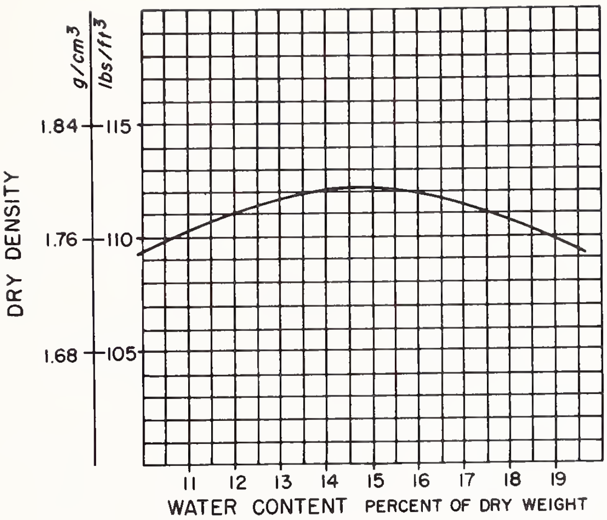


Figure 44. Moisture density relationship of the clay cap material. The American Society of Testing Materials has published and designated this procedure as D698, method A. Basically, the sample is air dried and placed into a 934 cm^3 mold for compaction by a 2.5 kg hammer with a 30.48 cm drop. The soil is poured into the mold one-third at a time and each layer receives 25 blows. Tests performed by Northern Testing Laboratories, Billing, Montana.

viously, the cap was constructed in two phases due to the limits of the dragline reach. Table 12 show the density-moisture characteristics of the first phase of the cap at various degrees of compaction. Normal scra-

Table 12. Statistical mean and standard deviation of clay cap bulk density* and moisture characteristics.

Depth (cm)	Dry Bulk Density (g/cm^3)	Water Content % H_2O by volume	Water Content % H_2O by weight
	Normal Scraper Operation; North Half of Cap		
0 - 20	1.69 ± .07	13.65 ± .96	8.08 ± .50
	Scraper Compaction, 2 Passes; North Half of Cap		
0 - 20	1.79 ± .09	14.2 ± 1.37	8.0 ± 1.04
	Final Scraper Compaction, 4-6 Passes; North Half of Cap		
0 - 20	1.79 ± .02	14.6 ± 1.22	8.1 ± .75
	Final Scraper Compaction, 4-6 Passes; South Half of Cap		
0 - 20	1.77 ± .06	16.7 ± 1.19	9.4 ± .89

---

*All measurements made with a Troxler 3411 density and moisture gauge with Am-Be and Cs-137 radioisotope sources per operation resulted in a dry density average of 1.65 g/cm^3 at an average 8.8% H_2O content on a weight basis. Two additional passes by loaded scrapers over the clayey material increased the average dry bulk density of the cap to 1.70 g/cm^3 (Table 12), but 4 to 6 total passes by loaded scrapers over the cap did not increase the density. The fact that the soil water content of 14.6% was not attained may have somewhat decreased the maximum obtainable density.

Our attempts to bring the clayey material to optimum water content for compaction with irrigation were partially successful. The pile was irrigated from the top which resulted in a water content gradient from about 20% at the surface to 7 and 8% a few feet from the surface.

Following completion of the first phase of the cap, the dragline began the covering process. Due to the possibility of rupturing the clay cap if the cover material was dumped with its usual force, a slower and more conservative method of deposition termed drift casting was used. The filled dragline bucket was brought in low across the cap and the material deposited in a manner resembling the unloading of a moving dump truck. This comparatively gentle application of cover material across the cap provided assurance that the integrity of the cap would not be destroyed. The described method was followed until 3 to 4 m of cover material had been deposited on the clay cap. The balance of the remaining overburden was used as cover material. Care was taken to minimize the amount of material deposited on the southern end of the cap so the second phase of cap construction could be tied to the first as effectively as possible. During this move it took about 12.5 hours to complete the stripping of the overburden.

The final 13 step move was made to the south early the morning of May 20. As it was no longer necessary to strip the surface 4.5 m selectively since the saline zone had diminished, work begin immediately to extend the basement for the second part of the capped area. When the basement fill material was built to a 15 m depth, the pit dozer began the leveling process as before. The dragline continued stripping overburden but this material went on top of the first phase of the capped study. The majority of the stockpiled saline material was deposited on the second part of the area to be capped. Following deposition and shaping of this material, the dragline was again forced to shut down while the second half of the cap was constructed. This shutdown was for about 5.0 hours.

Prior to transport to the capping area, the clayey material was at a moisture content of about 7.9% by weight and had an average dry density of 1.8 g/cm^3. Approximately 70 scraper loads of this clayey material, each 16.8 m^3 (22 yd^3) in volume, were needed to complete cap construction. Table 12 shows the density-moisture characteristics of the second phase of the cap. At 9.4% water content an average dry density of 1.77 g/cm^3 was obtained with 4 to 6 compaction passes by scrapers.

One point worthy of note is the overall manner of the cap construction. To insure that percolating soil moisture was effectively diverted around the buried saline material, it was necessary to umbrella the undesirable zone with the cap. The "umbrella-effect" was accomplished by centering the saline material on the basement bench and then dozing it to a 5:1 grade in all directions. The clayey material was then laid down over and out from the edges of the saline material (Figure 45).

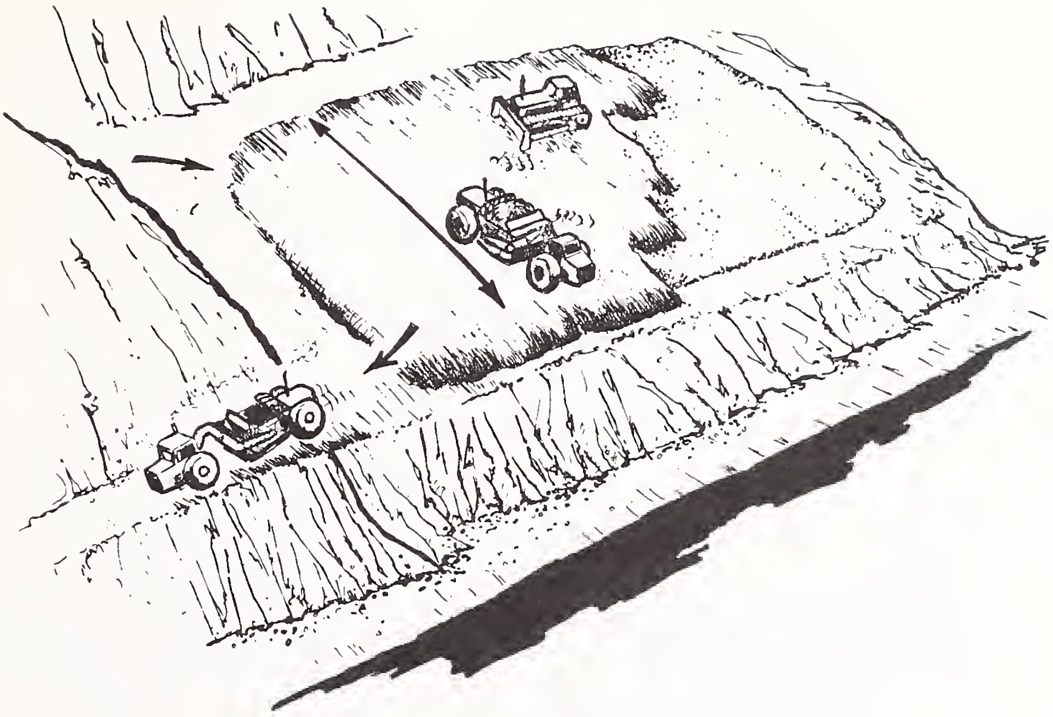


Figure 45. Loaded scrapers were used to compact the clay cap to bulk density of about 1.8 g/cm^3. A D-9 dozer shaped the cap to produce an "umbrella-effect" over the saline material.

In summary, about 2369 m^3 (3102 yd^3) of material with 50% clay content dominated by a kaolinite-illitic clay mineralogy were used to construct a clay cap over saline material. The dimensions and orientation of this cap are shown in Figure 46. Basically, the capped area extended the width of the pit (55 m), and along the pit for 62 m; resulting in an average cap thickness of 0.7 m, and a dry density of 1.78 g/cm^3.

With completion of the final phase of the clay cap, the dragline buried the zone using the drifting method described previously. A significant amount of overburden remained after the capped area was completely buried. All this material was spoiled over the demonstration study area (Figure 46). That part of the stockpiled saline material remaining on the S-5 highwall (about 1500 m^3) was carried down the highwall by the dragline as it continued to excavate the balance of Pass S-4 in a normal manner. At a point apprxoimately 60 m south of the demonstration study area, the saline material on the highwall was re-handled into the pit. The completed demonstration is shown in Figures 47 and 48.

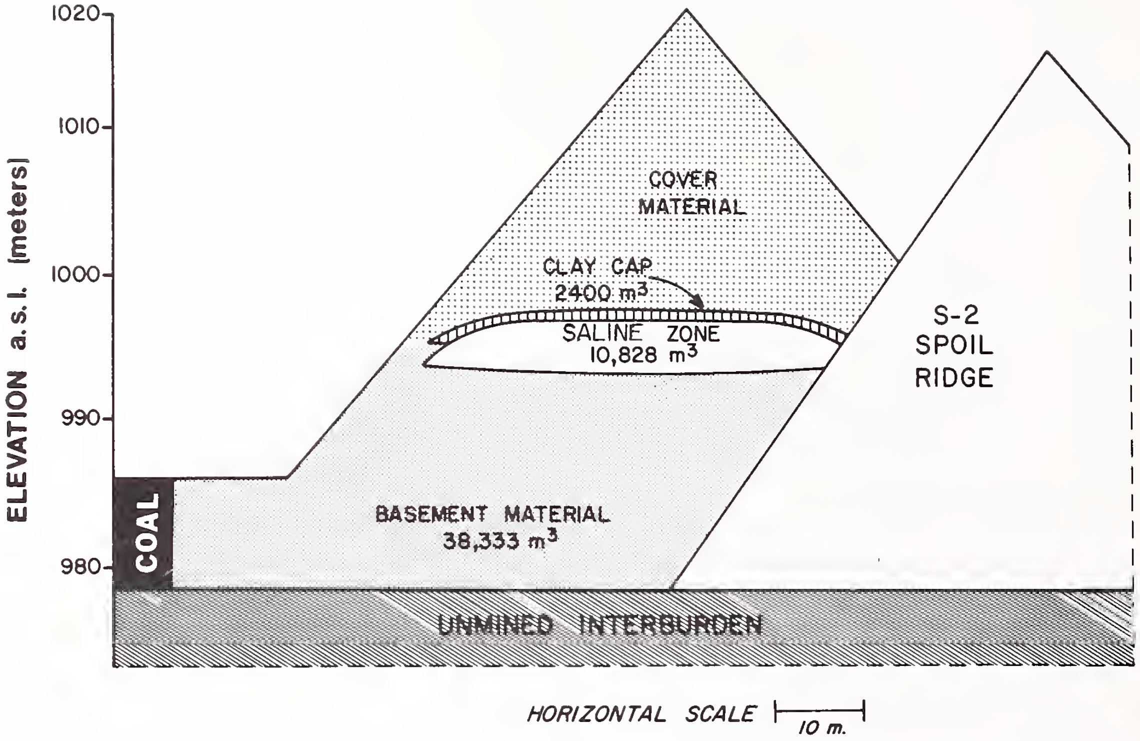


Figure 46. Transverse cross-section of the capped demonstration.

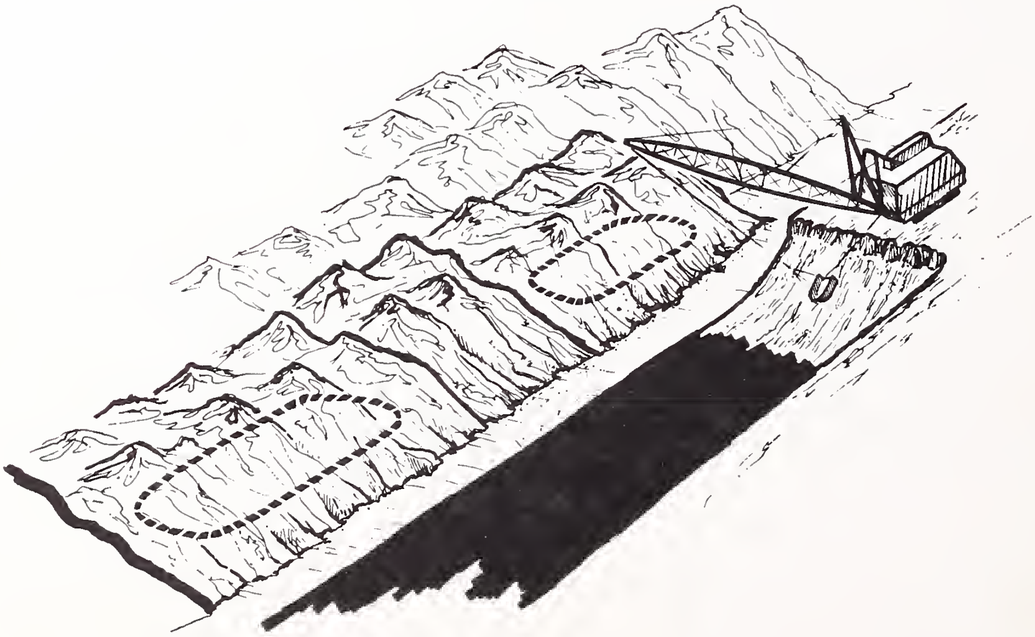


Figure 47. The uncapped and capped experiments were oriented as shown above at the conclusion of the demonstration.

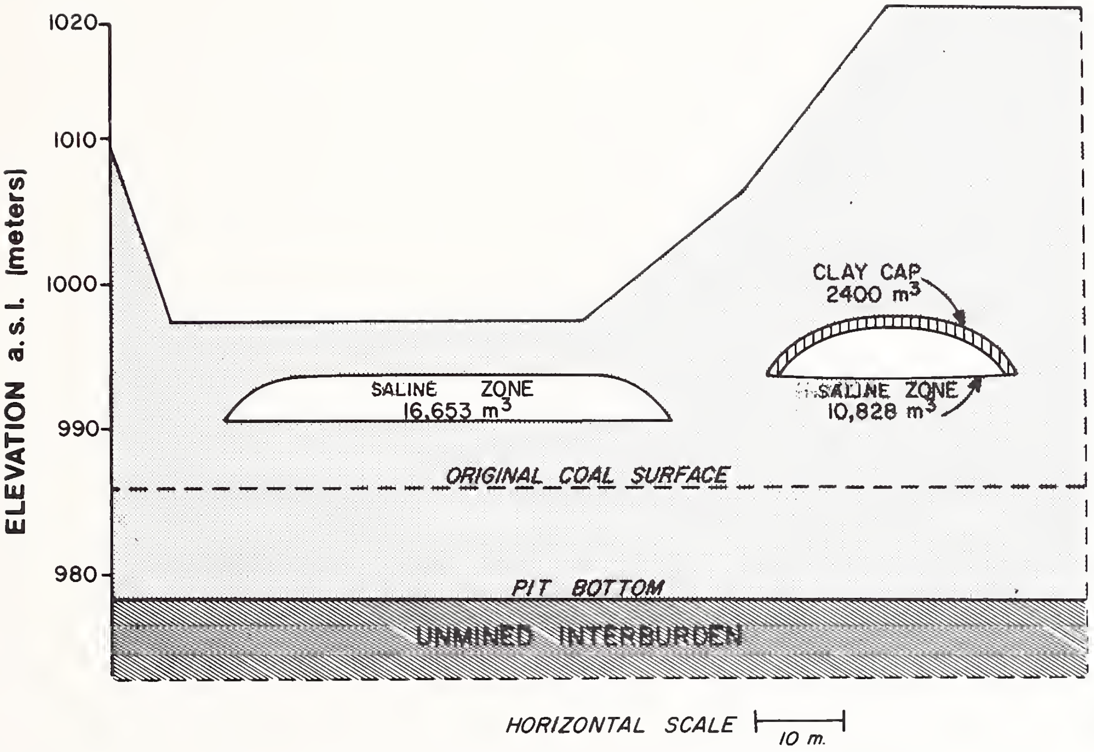


Figure 48. Longitudinal cross-section of the demonstration area May, 1977 before regrading.

ECONOMIC ANALYSIS

A comparitive economic analysis of normal mining costs versus the cost of selective overburden handling operations was undertaken following field demonstrations. The direct operating cost estimates included in this report reflect the cost of mining and reclamation for an average interior acre at the mine site. The added costs of highwall reduction and boxcut elimination were not considered. Tipple construction and operating costs were also excluded from the analysis.

Hourly equipment requirements for a normal operation were based on field observation and information supplied by the Western Energy Company and their mining contractor, Long Construction Company. Direct operating costs were estimated based on information from the aforementioned companies and equipment suppliers. The procedure for the analysis was to compare this estimate with the additional costs resulting from the selective handling operations. The additional costs data were supplied by Western Energy Company.

In order to perform this analysis the following general assumptions were made which reflect the overburden condition and physical arrangement of the mine in Area B. For reasons of clarity, this economic analysis generally presents English units rather than Metric.

- Consider one acre to have dimensions of 51.8 m x 78 m (170 ft x 256 ft).
- Average overburden thickness is 21.3 m (70 ft).
- Average topsoil thickness is 0.91 m (3 ft).
- Average coal thickness is 7.3 m (24 ft).
- One way distance between demonstration area and topsoil stockpile is 762 m (2500 ft).
- One way coal hauling distance to the tipple (including in-pit haul distance) is 4724 m (15,500 ft).
- Stripping ratio is 3:1.
- Overburden material is soft.
- Mining of a single coal seam which is horizontally oriented.
- Overburden and coal are fragmented by blasting prior to extraction.
- Pit floor condition is firm.
- Haul road conditions are good with maximum adverse grade of 5%.
- 42,450 metric tons (41,780 tons) of coal per acre.

On a per acre basis the uncapped study cost about $4515.00 more than the normal operation, and the capped study cost about $19,582.00 more than the normal operation (Table 13). These results can also be presented in terms of cost comparison ratios. The uncapped operation increased direct operating costs about 12%, while the capped operation increased the costs about 53%. The capped operation was about 36% more costly than the uncapped operation, due largely to the clay cap construction and associated shutdown time of the dragline.

Table 13. Summary of direct operating costs (1977) for the normal spoiling and selective handling operations.

	DIRECT OPERATING COST	
MINE OPERATION	PER ACRE	PER TON OF COAL
Normal Spoiling	$36,706.39	$ .88
Uncapped Study	$41,221,48	$ .99
Capped Study	$56,288.73	$1.35

- -

Uncapped operation to normal operation	1.12 : 1.00
Capped operation to normal operation	1.53 : 1.00
Capped study to uncapped study	1.36 : 1.00

We sincerely hope these results are not misleading. It must be realized that comparative costs for selective handling of material located at the base or middle of the overburden profile may be substantially different from those presented in this analysis. The location or availability of suitable clay for capping material could vary significantly and thus affect cost. Therefore, the merits of this analysis are site specific, and as the listed assumptions change so will economic analysis results.

The remainder of the discussion in this section presents the data base (Tables 14-20) used for the final results presented in Table 13. Direct operating costs for the normal mining operations (Table 14) were categorized into seven major steps, including topsoil removal, overburden preparation and removal, coal loading and hauling, regrading, and revegetation. It is interesting to note that the most expensive category was coal hauling followed closely by overburden removal. Direct operating cost for normal operation was about $36,706.00 per acre -- a cost that produced about 41,780* tons of coal. Table 20 shows the cost derivation for each of the seven major categories itemized in Table 14. When appropriate, references are stated in the cost derivation. Constants in the economic analysis such as physical dimensions and material volumes within the demonstration area are given in Table 16.

The direct operating cost for the uncapped study was $41,221.00/acre (Table 22), $4514.00 greater than normal operation. The source of this increased cost was from the categories of overburden removal, engineering-supervision, and foreman. The engineering-supervision and foreman costs were a result of the overburden removal work, and perhaps could have been included in this category. Consider, then, a comparison between direct operating costs in a normal operation and the uncapped study for only the overburden removal category, meaning that topsoil removal, overburden preparation, coal loading and hauling, regrading and backfill, and revegetation would not be in the comparison. These later categories are basically constants regardless of the way the overburden is handled. This reduces analysis to a cost comparison of spoiling techniques, and we find selective burial spoiling (uncapped study) was 1.4 times more costly than normal spoiling. These increased costs can be largely attributed to the extra D-9 dozer assigned to shape the spoiled material, additional supervision, and dragline delay and overburden rehandling. It must be realized, in order to place increased costs due to selective handling in perspective, that the overall economic analysis (Table 13) included all mining costs from topsoil removal to revegetation.

The direct operating cost for the capped study was $56,288.00/acre (Table 18), $19,582.00 greater than normal operation. The source of this increased cost was from the category overburden removal, and items H, I, and J (i.e., supervision, foreman, operators) which were a direct result of the overburden removal operation. Comparing, then, direct operating costs of overburden removal between the normal operation and the capped study, we find selective burial spoiling and capping (capped study) was 2.0 times more costly than normal spoiling. These increased costs can be largely attributed to dragline delay and rehandle, the extra D-9 dozer assigned to shape the spoiled material, and extra supervision.

---

*See assumptions

Table 14. Direct operating cost data for normal$^+$ mining operation. The analyses for each unit operation is given in Table 20.

Unit Operations	Quantity	Total Equip hrs/acre*	Total operator hrs/equip. type/acre*	DOC hourly charge	Total DOC/unit op.	Total DOC/acre*
A. Topsoil Removal						
D8 Push Cat	1	10.8	-	$40.50	$437.40	
Operator	1	-	10.8			
631C Scraper	3	32.5	-	43.54	1415.05	
Operator	3	-	32.5			
Subtotals	-	43.3	43.3		1852.45	$1852.45
B. Overburden Preparation						
BE60 Drill	1	17.5	-	47.60	833.00	
Operator	2	-	35.0			
D9 Dozer	1	1.0	-	47.25	47.25	
Operator	1	-	1.0			
Explosives Truck	1	8.0	-	16.00	128.00	
Operator	4	-	32.0	12.00	384.00	
Explosives	-	-	-	-	2709.35	
Subtotals	-	26.5	68.0	-	4101.60	4101.60
C. Overburden Removal						
M8050 Dragline	1	40.3	-	249.32	10047.60	
Operator	2	-	80.6			
D9 Nurse Cat	1	40.3	-			
Operator	1	-	40.3			
988 F-E Loader	1	7.6	-	39.82	302.63	
Operator	1	-	7.6			
Subtotals	-	88.2	128.5	-	10350.23	10350.23
D. Coal Loading						
BE 280B Shovel	1	41.8	-	67.82	2834.88	
Operator	1	-	41.8			
Failing Drill	1	8.0	-	18.00	144.00	
Operator	1	-	8.0			
Explosives Truck	1	6.0	-	16.00	96.00	
Operator	1	-	24.0	12.00	288.00	
Explosives	4	-	-	-	780.45	
Subtotals	-	55.8	73.8	-	4143.23	4143.23
E. Coal Hauling						
Euclid CH-120 Trucks	5	209.0	-	54.04	11294.36	
Operator	5	-	209.0			
Subtotals	-	209.0	209.0		11294.36	11294.36
F. Regrading & Backfill						
D8 Push Cat	1	11.7	-	40.50	473.85	
Operator	1	-	11.7			
631C Scraper	3	35.1	-	43.54	1528.25	
Operator	3	-	35.1			
D9 Dozer	2	53.7	-	47.25	2537.32	
Operator	2	-	53.7			
Subtotals	-	100.5	100.5		4539.42	$4539.42
G. Revegetation					425.00	
Subtotals	-	-	-		425.00	425.00
TOTALS		523.3	623.1			36706.39

* 1 acre = 2.47 hectares

+ See assumptions

Table 15. Unit operation economic analysis (1977) for topsoil removal, overburden preparation and removal, coal loading and hauling, regrading, and revegetation.

A. Topsoil Removal (removal and transport to stockpile)

1. Additional assumptions and calculations for equipment hour needs per acre.
   - a. Volume of topsoil per acre = 4835 yd^3.
   - b. Use 3 631c scrapers with one D8 push cat per acre.
   - c. Maximum capacity of a 631C scraper is 22 yd^3, assume 90% efficiency = 19.8 yd^3/haul
   - d. By field observation, it takes 8 minutes for one scraper to complete each round trip haul.
   - e. 60 minutes ÷ 8 minutes = 7.5 hauls/hr/scraper.
   - f. 7.5 hauls x 19.8 yd^3/haul = 148.5 yd^3/hr/scraper.
   - g. 3 scrapers x 148.5 yds/hr/scraper = 445.5 yds/hr for 3 scrapers.
   - h. 4835 yd^3/acre ÷ 445.5 yds/hr for 3 scrapers = 10.8 hrs to set up and stockpile one acre topsoil with 3 scrapers.
   - i. 3 scrapers x 10.8 hours = 32.5 scraper hours/acre.
   - j. In addition, one D8 push cat would be required for 10.8 hours.
2. Direct Operating Costs Estimates
   - a. Method to estimate DOC for D8 push cat (includes operator)
     M.A.E.S. Contractural Agreement with WECO*= $45.00/hr.
     ($45.00 + 15.00).675 = $40.50/hr
   - b. Method to estimate DOC for 631C scraper (includes operator)
     M.A.E.S. Contractural Agreement with WECO*= $49.50
     ($49.50 + 15.00).675 = $43.54/hr
3. Direct Operating Cost Per Acre Calculations
   - a. D8 push cat with operator- $40.50/hr x 10.8 hrs/acre = $437.40/acre
   - b. 631C scraper with operator-$43.54/ hr x32.5 hrs/acre = $1415.05/acre

B. Overburden Preparation

1. Additional assumptions and calculations for equipment hour needs per acre.
   - a. Normal blast hole drilling pattern consists of 4 rows per 170 ft. wide pass. Holes and rows are 35 feet apart.
   - b. Estimate 35 holes per acre.
   - c. Average 30 minutes to set up and drill each 70 foot hole.
   - d. 35 holes x 70 ft/hole = 2450 ft/acre of coal
   - e. 2450 ft ÷ 140 ft/hr = 17.5 hours of drilling
   - f. Construction of level surface for drill rig requires estimated 1 hr/acre with D9
   - g. 8 hours/acre required by explosives crew for overburden blasting
2. Direct Operating Cost Estimates
   - a. BE60 drill rig

Labor, 2 men (net)	$20.00	(27.12 gross)
Electric power	4.50	
Repairs, supplies, maintenance	10.50	
	$35.00/hour without bit cost ($.25/ft)	

Approximate bit cost $910.00 each
Estimated bit life (soft material) - 10,000 ft.
$910.00 ÷ 10,000 ft. = $.09/ft.
Estimated D.O.C. including bit cost
$.25 + $.09 = $.34/ft. drilled
140 ft/hr x .34/ft = $47.60/hour (includes labor and bits)

---

* M.A.E.S. Contractural Agreement + $15.00 was used to estimate total IOC + DOC. Factor of .675 x total cost is estimated DOC.

Table 15. (continued)

B. Overburden Preparation (cont.)
2. Direct Operating Cost Estimates
b. D9 Dozer (including operator)*
M.A.E.S. Contractural Agreement with WECO = $55.00/hr.
(55.00 + 15.00).675 = $47.25
c. Explosives truck with operators (source: Roundup Powder Co., Miles City and Long Construction Co., Colstrip)
Gross costs of explosives:
ANFO (NH_4 NO_3 + diesel fuel)(94% + 6%)
$8.35/100 lb (in bulk)
$11.60/100 lb (bagged)
Plastic hole lines:
$0.67/lb (50 lb rolls)
Primers:
$53.55/100 lb
Prima cord:

$36.40 per 1000 ft.

E - Cord:
$45.85 per 1000 ft.
For overburden blasting: (from Jim Rose)
Use 0.25 lb of ANFO per yd^3 of material to be blown.
If overburden drill hole spacing is 35 ft. x 35 ft. and the average hole depth is 70 ft. then 3176 yd^3 of material is blown per hole.
0.25 lb ANFO x 3176 yd^3 = 794 lbs ANFO/hole
Use 80 ft. of liner for 70 ft. hole at $0.08/ft of liner
Use 2 primers per hole
Use 72 ft. of prima cord per hole
Use 40 ft. of E-cord per hole
Cost of explosives per overburden hole
ANFO (bulk) 794 ÷ 100 x $8.35 = $66.30
Liner 80 ft. x 0.08/ft. = 6.40
Primacord 72 ft. ÷ 1000 x $36.40 = 2.62
E-cord 40 ft. ÷ 1000 x $45.85 = 1.84
Primers = .25
$77.41 total cost of explosives per overburden hole
Explosives truck at $16.00/hr without operator requires 4 man crew at $12.00/hr per crewman.
3. Direct operating cost per acre calculations
a. BE60 drill rig with bits and operators:
17.5 hrs/acre x $47.60/hr = $ 833.00/acre
b. D9 dozer with operator
1 hr/acre x $47.25/hole = 47.25/acre
c. Explosives
35 holes/acre x $77.41 per hole = 2709.35/acre
d. Explosives truck and crew
truck - 8 hrs x $16.00/hr = 128.00/acre
operators - 4 x $12.00 x 8 hrs. = 344.00/acre
$4061.60/acre

---

* M.A.E.S. Contractural Agreement + $15.00 was used to estimate total IOC+DOC. Factor of .675 x total cost is estimated DOC.

Table 15. (continued)

C. Overburden Removal
  1. Additional assumptions and calculations for equipment hour needs per acre.
    a. M8050 dragline
      7.5 hr/shift x 52 weeks/yr x 20 shifts/week = 7800 hrs/yr scheduled
      Use factor of 80% availability (accounts for unscheduled delays) = 6240 net digging hrs/year.
      Use 2800 net yd^3/hr (Longs Construction data based on pass S-2 productions, see Table 25).
      6240 hrs/yr x 2800 yd^3/hr = 17,472,000 yd^3/year.
      170 x 256 x 70 ÷ 27 ft^3/yd = 112,830 yd^3/acre; 112,830 yd^3/acre ÷ 2800 yd^3/hr = 40.3 hrs/acre
    b. D9 nurse cat (included with M8050 calculations)
    c. 988 front end loader - estimated 7.6 hrs/acre
  2. Direct operating cost estimates
    a. M8050 dragline
      at $0.125/yd^3 (figure supplied by WECO)
      17,472,000 yd^3/year x $0.125 = $2,184,000/year in D.O.C.
      Based on 8760 gross hours/year, the per hour D.O.C.
      $2,184,000/yr ÷ 8760 hrs/year = $249.32/hr (includes D9 nurse cat and operator)
      Information from Marion Company and Bucyrus-Erie Company indicated the D.O.C./yd^3 at $0.10. Both companies advised this figure would include the nurse cat operator but not the cat itself.

      The M8050 handbook indicated on page 3 of appendix C (Dragline Data Book, February, 1976) is the same machine as in Area B.
        $210.06 x 1.2 to allow for cost escalations since 1975 = $252.07
      Add $37.00 for operating cost of D9 cat without operator = $289.07
      Considering the difference in estimated productions (2800 yd^3/hr vs. 2145 yd^3/hr, the agreement is reasonable.
    b. D9 nurse cat (included in dragline cost)
    c. 988 front end loader
      (44.00 + 15.00).075 = $39.82/hr*
  3. Direct operating cost per acre calculations
    a. M8050 dragline
      $249.32/hr x 40.3 hrs/acre = $10,047.60/acre for M8050 and operators and D9 nurse cat with operator
    b. D9 nurse cat - included with M8050 hourly D.O.C.
    c. 988 front end loader - 7.6 hrs. x $39.82/hr = $302.63/acre

D. Coal Loading
  1. Additional assumptions and calculations for equipment hour needs per acre.
    a. Volume of coal per acre = 24 ft x 170 ft x 256ft/27 ft^3/yd^3 = 38,685 yd^3
    b. 1.08 tons/yd^3 x 38,685 yd^3 = 41,780 tons/acre
    c. BE 280-B can load 1000 tons per hour using a 5 truck spread (from Long Construction Co., Colstrip).
      All trucks are CH-120 Euclids and one way haul distance is 15,500 ft.
      It requires 41.8 digging hours to load one acre of coal.

---

* M.A.E.S. Contractural Agreement + $15.00 was used to estimate total IOC+DOC. Factor of .675 x total cost = D.O.C.

Table 15. (continued)

D. Coal Loading (cont.)
  d. 8 hrs required for Failing rotary rig to drill coal blast holes on 22 ft x 22 ft grid on one acre of coal (55 holes/acre)
  e. Coal blast holes are 24 ft. deep
  f. 430 yd^3 of material blown per hole
  g. 6 hrs required for explosive trucks and crew per acre
 2. Direct operating cost estimates
  a. BE-280-B (information from B-E)

Operating crew (2 men)	$27.12/hr	
Electric power	8.80/hr	
Maintenance, supplies, repairs	31.90/hr	
	$67.82/hr	D.O.C.

  b. Failing rotary rig - $18.00/hr with operator
  c. Explosives
   108 lbs ANFO(bagged)per hole x $11.60/100 lbs. = $12.53
   26 ft of primacord x $36.40/1000 ft = .95
   10 ft of E-cord x $45.85/1000 ft = .46
   2 primers = .25
   no liner = .00
   $14.19 explosives/coal blast hole

  d. Explosives truck and crew
   Truck - $16.00/hr without operator
   Crew - 4 crew members $12.00/hr per crewman
 3. Direct operating cost per acre calculations
  a. BE-280-B
   $67.82/hr x 41.8 hrs/acre = $2834.88/acre
  b. Failing rotary rig
   $18.00/hr x 8 hrs/acre = 144.00/acre
  c. Explosives
   $14.19/hole x 55 holes/acre = 780.45/acre
  d. Truck and crew
   $16.00/hr x 6 hrs. = 96.99/acre (trucks)
   $12.00/hr x 4 operators x 6 hrs. = 288.00/acre (operators)
   $4143.33/acre total D.O.C. for coal loading

E. Coal Hauling
 1. Additional assumptions and calculations for equipment hour needs per acre.
  a. Production based on 5 trucks spread using Euclid CH-120 haul trucks
  b. Use 41.8 hours to haul 1 acre of coal to tipple
  c. 5 trucks/acre x 41.8 hours/acre = 209.0 truck hours/acre
 2. Direct operating cost estimates (from Euclid)
  (Euclid figures escalated by 1.2 and operator and labor costs adjusted to bring into line with WECO operation)

Tire replacement cost	$10.19
Tire repairs	1.02
Repairs	17.17
Fuel	7.68
Preventive maintenance	4.42
Operator	13.56
	$54.04/hr

Table 15. (continued)

E. Coal Hauling (cont.)
    3. Direct operating cost per acre calculations
        $54.04/hr x 209 hrs/acre = $11,294.36/acre D.O.C.

F. Regrading and Backfill
    1. Additional assumptions and calculations for equipment hour needs per acre.
        a. Regrading and backfilling includes:
            Redistribution of topsoil onto the regraded spoils
            Regrading of spoils to required contour
            Establishment of acceptable drainages and elimination of depressions
        b. Regrading and contouring was done using two D9 dozers
        c. Redistribution of topsoil and filling of depressions was accomplished with a 3 scraper spread of 631C scrapers and a D8 push cat
        d. One D9 dozer can move 300 yd^3 per hour with a maximum push of 500 feet (from Long Construction Co.)
        e. Estimate that 10 ft of spoil material was displaced in the regrading.
        f. 170 ft x 256 ft ÷ 27 ft^3/yd^3 = 16,120 yd^3/acre
        g. 16,120 yd^3/acre ÷ 300 yd^3/hr = 53.7 D9 hrs/area for regrading
        h. Estimate 35 total scraper hours to redistribute topsoil and fill depressions
            32.5 hours from Unit Operation A
            2.5 hours for depression fill, etc.
            35.0 total scraper hours
    2. Direct operating cost estimates
        a. D9 dozer*
            ($55.00 + $15.00).675 = $47.25/hr (includes operator
        b. 631C scraper*
            (49.50 + 15.00).675 = $43.54/hr (includes operator)
        c. D8 push cat*
            ($45.00 + $15.00).675 = $40.50/hr (includes operator)

    3. Direct operating cost per acre calculations
        a. D9 dozer
            $47.25/hr x 2 dozers x 26.85 hrs/dozer = $2537.33/acre
        b. 631C scrapers
            $43.54/hr x 3 scrapers x 11.7 hours/scraper = 1528.25/acre
        c. D8 push cat
            $40.50/hr x one D8 x 11.7 hours = 473.85/acre
            $4539.43/acre

G. Revegetation
    Figure a cost of $425.00/acre which includes:
        seeding
        mulching
        fertilizing
        miscellaneous
    (Based on data supplied by reclamation supervisor, WECO)

---

* M.A.E.S. Contractural Agreement + $15.00 was used to estimate total cost (IOC + DOC). Factor of .675 x total cost = estimated D.O.C.

Table 16. Physical dimensions and material volumes within the demonstration area.

a. Land area of the demonstration site:

Uncapped Study	$\frac{170 \text{ ft} \times 195 \text{ ft}}{43{,}560 \text{ ft}^2/\text{acre}}$ =	0.76 acres	43.2%
Buffer Zone	$\frac{170 \text{ ft} \times 100 \text{ ft}}{43{,}560 \text{ ft}^2/\text{acre}}$ =	0.40 acres	22.7%
Capped Study	$\frac{170 \text{ ft} \times 155 \text{ ft}}{43{,}560 \text{ ft}^2/\text{acre}}$ =	0.60 acres	34.1%
Total Land Area	=	1.76 acres	100.0%

b. Volume of overburden material removed:

Uncapped Study	86,845 yd^3
Buffer Zone	45,865 yd^3
Capped Study	68,631 yd^3
Total	201,341 yd^3

c. Volume and tons of coal produced within the demonstration area:

Uncapped Study	29,467 yd^3	x 1.08* =	31,824 tons
Buffer Zone	15,111 yd^3	x 1.08 =	16,320 tons
Capped Study	23,422 yd^3	x 1.08 =	25,296 tons
Totals	68,000 yd^3		73,440 tons

* 1.08 conversion factor used for yd^3 to tons of coal

Tables 19 and 20 contain baseline economic analysis data used in the economic analysis. Actual costs in excess of the normal mining operation due to selective handling operations are categorically summarized in Table 19. The increased costs associated with the 8050 dragline are of particular interest. Lost production in cubic yards is indicated for the uncapped and capped studies, and for the buffer zone construction. The loss in production was based on one very important statistic, the average dragline production rate (Table 20). This statistic was developed with the 8050 dragline production rates from mine Pass S-2, which was located adjacent to the pass containing the study area (Pass S-4, see Figure 25). Since overburden conditions were very similar in these two passes, the average production rate of 2800 yd^3/hr developed in Pass S-2 was used for determining lost production in Pass S-4 due to selective handling studies.

Table 17. Comparison between direct operating cost (1977) in a normal* operation and the uncapped study demonstration.

	Time/.76 acres "normal"(HRS)	Cost/.76 acres* normal	Additional time/.76 acres uncapped (HRS)	Additional cost/.76 acres uncapped	Total time/ .76 acres uncapped (HRS)	Total DOC/ .76 acres uncapped	Total time/ 1 acre uncapped (HRS)	Total cost/ 1 acre uncapped
A. Topsoil Removal								
D8 Push Cat	8.2	$332.40	-	$ 0	8.2	$ 332.40	10.8	$ 437.40
631C Scraper	24.7	1075.40	-	0	24.7	1075.40	32.5	1415.05
B. Overburden Preparation								
BE60 Drill	13.3	633.08	-	0	13.3	633.08	17.5	833.00
D9 Dozer	0.8	35.91	-	0	0.8	35.91	1.0	47.25
Explosives Truck	6.1	97.28	-	0	6.1	97.28	8.0	128.00
Explosives Crew	24.32	291.84	-	0	24.32	291.84	32.0	384.00
Explosives	-	2059.11	-	0	-	2059.11	-	2709.35
C. Overburden Removal								
M8050 Dragline	30.6	7636.18	1.1	269.27	31.7	7905.45	41.7	10,396.64
D9 Nurse Cat	-	0	-	0	-	0	-	0
988 F-E Loader	5.8	230.00	-	0	5.8	230.00	7.6	302.63
D9 Pit Dozer	-	0	38.5(1)	1819.13	38.5	1819.13	50.7	2395.58
D9 Push Cat	-	0	-	0	-	0	-	0
631C Scraper	-	0	-	0	-	0	-	0
14G Grader	-	0	-	0	-	0	-	0
D. Coal Loading								
BE 280B Shovel	31.8	2154.51	-	0	31.8	2154.51	41.8	2834.88
Failing Drill	6.1	109.44	-	0	6.1	109.44	8.0	144.00
Explosives Truck	4.6	72.96	-	0	4.6	72.96	6.0	96.00
Explosives Crew	18.2	218.88	-	0	18.2	218.88	24.0	288.00
Explosives	-	593.14	-	0	-	593.14	-	780.45
E. Coal Hauling								
Euclid CH-120 Trucks	158.8	8583.71	-	0	158.8	8583.71	209.0	11,294.36
F. Regrading and Backfill								
D8 Push Cat	8.9	360.13	-	0	8.9	360.13	11.7	473.85
631C Scraper	26.7	1161.47	-	0	26.7	1161.47	35.1	1528.25
D9 Dozer	40.8	1928.36	-	0	40.8	1928.36	53.7	2537.32
G. Revegetation		323.00	-	0	-	323.00	-	425.00
H. Engineering & Supervision			28.5+7.5+8.5=44.5(4)	1042.19	44.5	1042.19	58.6	1371.30
I. Foreman			16.0 reg.(2)	222.08	16.0	222.08	21.0	291.48
			4.0 o.t.(3)	81.29	4.0	81.29	5.3	107.69
TOTALS		27,896.80		3433.96		31,330.76		41,221.48

(1) - (4) see WECO cost distribution sheet (Table 19)

* See assumptions (page 80).

Table 18. Comparison between direct operation cost (1977) in a normal* operation and the capped study demonstration.

	Time/.60 acres "normal" (HRS)	Cost/.60 acres normal	Additional time/.60 acres capped (HRS)	Additional cost/.60 acres capped	Total Time/ .60 acres capped (HRS)	Total DOC/ .60 acres capped	Total time/ 1 acre capped	Total cost/ 1 acre capped
A. Topsoil Removal								
D8 Push Cat	6.5	$ 262.44	0	$ 0	6.5	$ 262.44	10.8	$ 437.40
631C Scraper	19.5	849.03	0	0	19.5	849.03	32.5	1415.05
B. Overburden Preparation								
BE60 Drill	10.5	499.80	0	0	10.5	499.80	17.5	833.00
D9 Dozer	0.6	28.35	0	0	0.6	28.35	1.0	47.25
Explosives Truck	4.8	76.80	0	0	4.8	76.80	8.0	128.00
Explosives Crew	19.2	230.40	0	0	19.2	230.40	32.0	384.00
Explosives	-	1625.61	0	0	-	1625.61	-	2709.35
C. Overburden Removal								
M8050 Dragline	24.2	6028.56	16.7(15)	4163.64	40.9	10,192.20	68.2	16,987.00
D9 Nurse Cat	-	0	-	-	-	-	-	-
988 F-E Loader	4.6	181.58	2.0(12)	79.64	6.6	261.22	11.0	435.36
D9 Pit Dozer	-	0	-	-	-	-	-	-
D9 Push Cat	-	0	16.0(10)	756.00	16.0	756.00	26.7	1260.00
631C Scraper	-	0	0	0	0	0	0	0
14G Grader	-	0	4.0(13)	159.30	4.0	159.30	6.7	265.50
D. Coal Loading								
BE 280B Shovel	25.1	1700.93	0	0	25.1	1700.93	41.8	2834.90
Failing Drill	4.8	86.40	0	0	4.8	86.40	8.0	144.00
Explosives Truck	3.6	57.60	0	0	3.6	57.60	6.0	96.00
Explosives Crew	14.4	172.80	0	0	14.4	172.80	24.0	288.00
Explosives	-	468.27	0	0	-	468.27	-	780.45
E. Coal Hauling								
Euclid CH120 Trucks	125.4	6776.62	0	0	125.4	6776.62	209.0	11,294.36
F. Regrading & Backfill								
D8 Push Cat	7.0	284.31	0	0	7.0	284.31	11.7	473.85
631C Scraper	21.1	916.95	59.0(11)	2568.86	80.1	3485.81	133.5	5809.68
D9 Dozer	32.2	1522.39	57.0(9)	2693.25	89.2	4215.64	148.7	7026.07
G. Revegetation	-	255.00	0	0	-	255.00	-	425.00
H. Engineering & Supervision	-	-	22.5(14)	526.95	22.5	526.95	37.5	878.25
I. Foreman	-	-	reg. 24.0(6)	333.12	24.0	333.12	40.0	555.20
	-	-	o.t. 8.0(6)	162.56	8.0	162.56	13.3	270.93
J. Operators for Claywork	-	-	op. 13.0(7)	176.28	13.0	176.28	21.7	293.80
	-	-	lab 10.0(8)	129.80	10.0	129.80	16.6	216.33
TOTALS		22,023.84		11.749.40		33,773.24		56,288.73

(5) - (15) see WECO cost distribution sheet (Table 19)

* See assumptions (page 80).

Table 19. Distribution of costs (1977) above normal mining operation due to the uncapped and capped selective handling operations.

OPERATION and DATE		FORMAN $13.88-REG. $20.32-O.T.	OPERATOR $13.56	LABORER $12.98	D9 DOZER $55	D9 PUSH CAT $55	631 SCRAPER $49.50	988 LOADER $44	14G MOTOR PATROL $35	ENG. SUP. $23.42	8050 DRAGLINE and D9		CLAY CREDIT	TOTAL COST
PRE-ENG.														
4/25										3.5				
4/26										6.0				
4/28										3.5				
5/13										2.0				
	Total Hrs									15.0 (4)				
	Cost									$351.30				$351.00
UNCAPPED SUTDY												Thus, 32.1 hrs operation 2800 yd^3/hr ave. production therefore, 89,880 yd^3 normal prod. 86,945 yd^3 actual prod. 3,035 yd^3 lost production		
5/16		8-Reg.			14.5					16.5	12.9			
		4-O.T.												
5/17		8-Reg.			24.0					12.0	18.2			
5/18											1.0			
	Total Hrs	16 Reg/4 OT			38.5 (1)					28.5 (4)	32.1			
	Cost	$303.37			$2117.50					$667.47				$3,467.72
BUFFER ZONE												Thus, 18.1 hrs operation 50,680 yd^3 normal prod. 45,865 yd^3 actual prod. 4,815 yd^3 lost production		
5/17		4-O.T.									4.4			
5/18		4-Reg.			14					13.5	13.7			
	Total Hrs	4 OT/4 Reg			14					13.5	18.1			
	Cost	$136.81			$770.00					$316.17		$601.88 @ .125¢/$yd^3$		$1,824.86
CLAY CAP STUDY												Thus, 41.2 hrs operation 115,360 yd^3 normal prod. 68,631 yd^3 actual prod. 46,729 yd^3 lost production $\frac{46{,}729\ yd^3}{2{,}800\ yd^3}$ = 16.7 hrs lost (15)		
5/4					1			2						
5/11		1	3	6										
5/12		3	9	2										
5/13			1	2										
5/18		4			8	4	16				7.6			
5/19		4-O.T.												
		8-Reg.			24	7	23			13.5	16.1			
5/20		4-O.T.												
		8-Reg.			24	5	20		4	9	17.5			
	Total Hrs	24 Reg/8 OT (6)	13 (7)	10 (8)	57 (9)	16 (10)	59 (11)	2 (12)	4 (13)	22.5 (14)	41.2			
	Cost	$495.72	$176.28	$128.80	$3135.00	$880.00	$2920.50	$88.00	$140.00	$526.95		$5841.13 @ .125¢/$yd^3$		$13,333.38
REMOVE EXCESS CLAY FROM PIT										The overburden zone of clay capping material was saved for the demonstration and not spoiled. To much was saved and the excess was removed. Since this constituted overburden removal of 10,300 yd^3, a credit was allowed.				
5/31		3-Reg.				6	17							
6/1		10-Reg.				20	65							
	Total Hrs	13 Reg.				26	82							
	Cost	$180.45				$1430.00	$4059.00					(Credit $1,339)		$4,330.45
POST ENG. ANALYSIS														
6/8										3				
6/10										5				
6/3										9				
	Total Hrs									17				
	Cost									$398.14 (4)				$398.14

*Average dragline production rate (see Table 20)

+References in Tables 17 and 18

Table 20. Summary of the dragline operations and production rates for the excavation of Pass S-2 (Provided by Long Construction Company).

Date	Overburden Produced yd^3 per day	Overburden Produced yd^{3a} per hour	Operation[b] Time, hrs	Controllable Delays, hrs	Maintenance Delays, hrs	Operating Delays, hrs	Operating Time Plus Operating Delays, hrs
2-25	56,098	2737	17.4	6.6	3.5	3.1	20.5
2-28	74,897	3119	21.3	2.7	0.0	2.7	24.0
3-1	63,297	2986	19.2	4.8	2.8	2.0	21.2
3-2	66,797	2783	20.7	3.3	0.0	3.3	24.0
3-3	49,330	2881	14.0	10.0	8.7	1.3	15.3
3-4	66,397	2767	21.2	2.8	0.0	2.8	24.0
3-7	63,598	2706	20.1	3.9	0.5	3.4	23.5
3-8	66,498	2770	21.4	2.6	0.0	2.6	24.0
3-9	60,697	2822	19.3	4.7	2.5	2.2	21.5
3-10	43,199	2787	13.7	10.3	8.5	1.8	15.5
3-11	61,097	2645	18.8	5.2	0.9	4.3	23.1
Average	60,607	2817	18.8	5.1	2.5	2.7	21.5

a) - Total yd^3 per day divided by operation time plus operating delays
b) - Time spent in actual excavation

COMMENTS

Under the guidance of the M.A.E.S. the demonstration was engineered by Long Construction Company, the mining contractor in Area B for the Western Energy Company, At our request, the following comments were prepared by the mining engineer from Long Construction Company who was assigned to this research project, and we quote:

Briefly, the idea of selective spoil placement is to create a situation in a spoil pile where a saline or toxic soil zone from the overburden would be placed in a position in the spoils so it would not adversely affect the ground water or reestablished surface vegetation. The purpose of placing a clay cap above the buried saline material is to create an impervious barrier to waters migrating upward or downward, thus inhibiting the movement of undesirable ions from the saline material.

In my opinion, a large walking dragline is not well-suited to the detailed work required in this type of operation. Equipment of this type is ideally suited to a large volume operation where a simple repetitious maneuver can be made. When operating conditions require a deviation from a normal operation the inflexibility of the dragline shows up as low hourly production rates and ultimately higher production costs. A break down of the operating problems are as follows:

1) Pit Configuration

The pit conditions during the selective handling demonstration were ideal. The pass being stripped and the pass being spoiled into were both 52 m (170 ft) wide which we feel is close to the optimum pass width. The overburden depth of approximately 23 to 24 meters is well within the easy spoiling depth for the machine. Extended bench operations are not required with the machine used until overburden is in excess of 30 m (100 ft). Overburden depths of under 15 m (50 ft) cause minor problems with land recontouring as overburden swell is not sufficient to offset the volume of the coal removed. The pit in the study zone was straight and was not complicated with either inside or outside curves.

In summarizing the operating conditions, they can only be characterized for the study as ideal. Normal pit operations may well require the stripping unit to operate with all of the preceding stated problems, any of which may greatly complicate the operation.

2) Overburden Stratigraphy

The saline material zone during the study was the surface 4.5 to 6 meters. This zone was easily recognized by the operator and could be separated from other material with a great deal of confidence. In cases where the saline material is located in the middle or the base of the overburden, the operator may find it virtually impossible to distinguish the saline zone. Weather and night operating conditions can further complicate this problem significantly. Dragline buckets usually make a pass cutting upward through the overburden and not in a level plane. This upward cutting action may require a 1.5 m (5 ft) saline zone to be blended with 1.5 m of material above and below the saline zone to insure its removal. This would then mean a narrow 1.5 meter zone would require treatment as though it was much larger. As a matter of scale, 1.5 meters is approximately one-half the depth of a 46 m^3 (60 yd^3) bucket.

3) Selective Handling System

The design of the system used for the selective handling demonstration was made with the idea that the saline zone could be of infinite length and yet the system would be compatible with normal operations for areas where saline material did not exist.

The operation in the demonstration area, for example, would have been simplified if the dragline tub had been placed on the elevation of the base of the saline zone. This position would allow the saline material to be chop-dug in advance of the dragline and would eliminate the rehandling of that material. This approach was not used as it would require ramping the dragline down to that level at the start of the study area and then back up at the end of the area. By keeping the dragline at the overburden surface elevation the selective spoiling demonstration was then campatible with normal operations.

4) Operational Delays

One of the major costs involved with selective spoil placement with a dragline is equipment delays due to the interactivity of several pieces of equipment in a confined work area. This problem can be easily seen where the dragline was idled so the scrapers could place and compact the impervious cap. Other dragline operations were adversely affected but not to the extent that was noted above.

5) Engineering and Supervision

Selective spoil placement operations would require individual engineering study. Additional supervision and coordination in the field would be required during the operation.

6) Tie In With Adjacent Passes

In areas where the saline zone extends through several passes, continuation of the impervious cap from one pass to the next may be very difficult. Making a good seal between passes would be very important as the outside edge of an old spoil slope may make a natural water channel between the spoil ridges. This would allow downward moving water to flow freely through the saline material in the spoils.

7) Spoil Pile Subsidence

In areas where wet bottom conditions exist and the spoils are piled against a coal fender, removal of the main coal body may allow the

> *toe of the spoil to slide into the pit. Subsidence of this type could easily rupture a clay cap and also tilt it so as to create a water channel through the saline material. This condition is the exact opposite to the condition that construction of the cap was trying to create.*

The following comments are based on observation by M.A.E.S. personnel of mining procedures both prior to and during the selective handling demonstration.

A review of the films taken during the demonstration and of the field notes indicates that refinements in the procedures used could reduce the overall production loss and thereby improve the cost:benefit ratio. The two primary areas where modifications would reflect the most gain are in the re-handling of saline material and dragline downtime. Reductions in time spent in these two areas would have the most positive effect on increasing net production.

Dragline downtime might be reduced in several ways. Maintaining spoil deposition within the boom radius would be a major step. One of the main reasons for shuttting down the dragline was in the interest of safety for swing in either direction. If equipment is working in the pit bottom (i.e., leveling and distributing deposited saline material), the dragline could swing the long way and continue building the basement bench. By going the long way, the bucket would never cross the lower work area. One objection to a long swing was the possible danger to the nurse cat. However, if the nurse cat was not actually working on pad construction or trimming the highwall (and it often wasn't), it could be pulled back out of the boom radius. Although the long swing is obviously more time consuming than a short swing, it would still be better than shutting down completely. There were also times when the nurse cat could have been used to assist the pit dozer. Use of the nurse cat in the pit bottom, while not always necessary, could reduce grading and preparation by about 50% at times.

The initial deposition of saline material on the highwall was unavoidable. At least that quantity, and perhaps a similar second volume of surface material, would have to be rehandled in any event. If the basement bench were built up higher than the 12 m used in the demonstration, more material could have been deposited within the effective working radius of the boom. This modification might tend to alleviate the "dirt-bound" situation noted previously plus increase the elevation of the bottom of the buried saline material.

In summary, it is apparent selective handling of overburden zones will increase the cost of the mining operation. The increased cost will be a function of many variables and the results presented here have to be considered a site specific example. Certain questions were not addressed regarding the economic impact of such increased costs on the mining industry. The purpose of this research was to develop selective handling technqiues and document the costs. Hopefully this study and others can be used to evaluate the balance between mining costs and environmental concerns.

VII

SUBSURFACE MINE SPOIL HYDROLOGY OF SELECTIVELY BURIED OVERBURDEN MATERIAL

One mine land reclamation concept suggests undesirable overburden materials which have been selectively buried during backfilling should be capped with a relatively impermeable layer to hinder the leaching of chemical constituents into nearby aquifers. Such a procedure would fulfill the intent of the Federal Reclamation Regulations which states *all toxic forming materials shall be selectively buried during backfilling, and where advisable, compaction methods shall be implemented to prevent leaching of toxic materials* (O.S.M., 1977).

In order to test this reclamation concept a zone of saline overburden material was selectively buried in May, 1977 during the dragline backfilling process at an intermediate depth in the spoil pile so neither the root zone nor the reestablished aquifer would be intercepted. A portion of this buried saline material was also capped with a 1 m thick layer of clay and compacted. This chapter reports on the subsurface hydrology associated with these selectively buried materials.

Since the *in situ* clay cap system was buried under more than 12 m of spoil and the annual precipitation at the research site averaged only 38 cm per year, it was anticipated that at least several years would be required before appreciable soil water levels might develop around the cap. Therefore a 1/20 scale version of the cap system was constructed to test the effectiveness of the clay cap design.

METHODS

All research was conducted near Colstrip, Montana at the Western Energy Company Rosebud Mine. An earlier chapter described the selective overburden handling sequence used in the experimental area (see Chapter VI). Soil water content was determined with the neutron scattering method (Schultz, 1966; McHenery, 1963) at various locations both above and around buried saline materials (Figure 49). Boreholes for the aluminum access tubing were drilled with a rotary table drill rig using a 5 cm (2 inch) drill bit. The 12 m (40 ft) sections of 5 cm diameter aluminum tubing were coupled with a combination of plastic heat shrink tubing over which a 15 cm long copper pipe was placed to add rigidity to the junction.

In order to improve accuracy of the neutron probe technique beyond that offered by the factory supplied calibration, special site specific calibrations were developed (Figure 50). When imperfect junctions between 12 m lengths of tubing prevented passage of the 5 cm diameter probe, a 3.8 cm diameter probe was used. The larger distance between the tube wall and the 3.8 cm diameter probe, compared to the 5 cm diameter probe, does not introduce significant error (Abeele, 1978) as long as *in situ* calibration is performed (Figure 50). Specific calibration procedures are described in Appendix E.

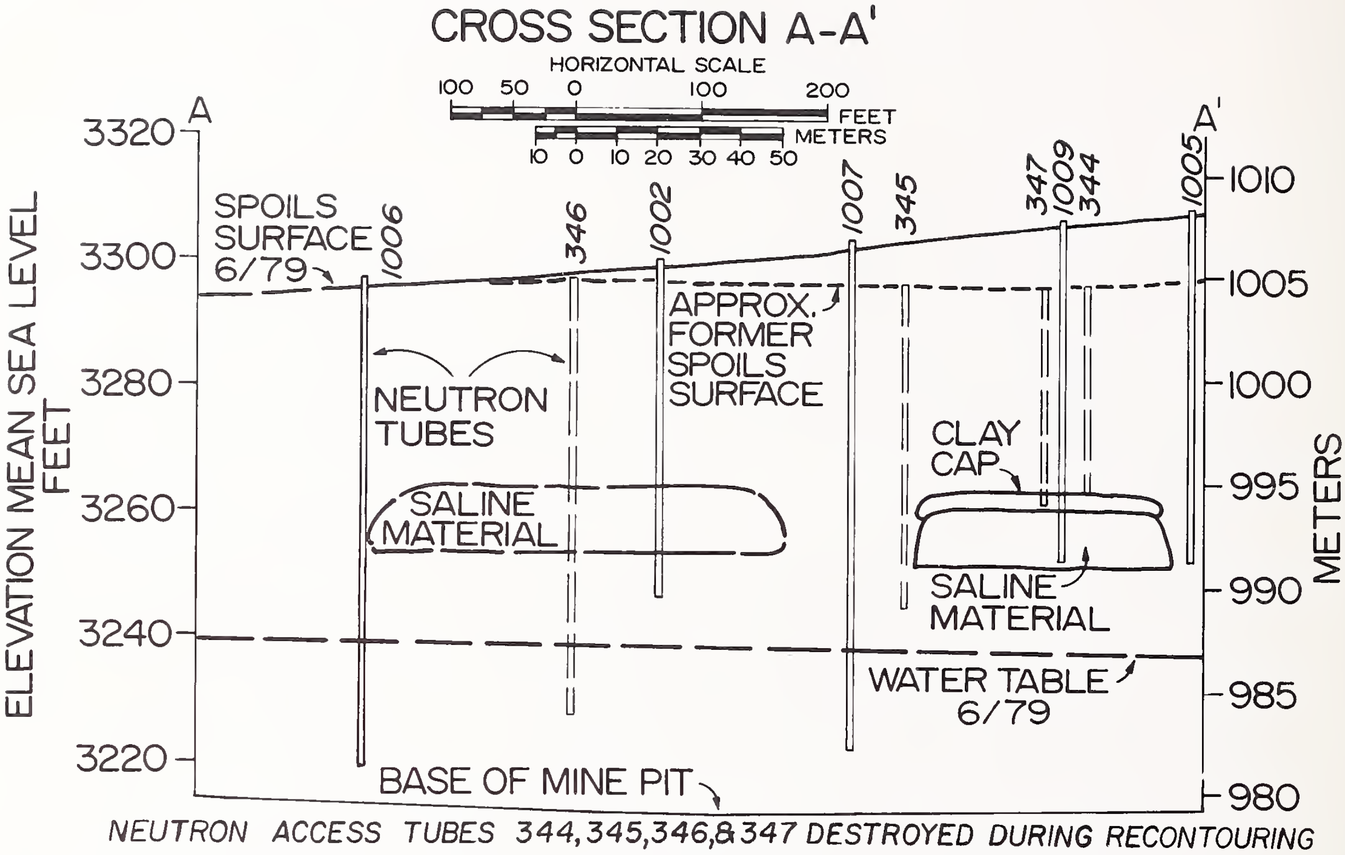


Figure 49. Experimental design of neutron tube position in reference to buried saline material. See Figure 53 for location of cross section.

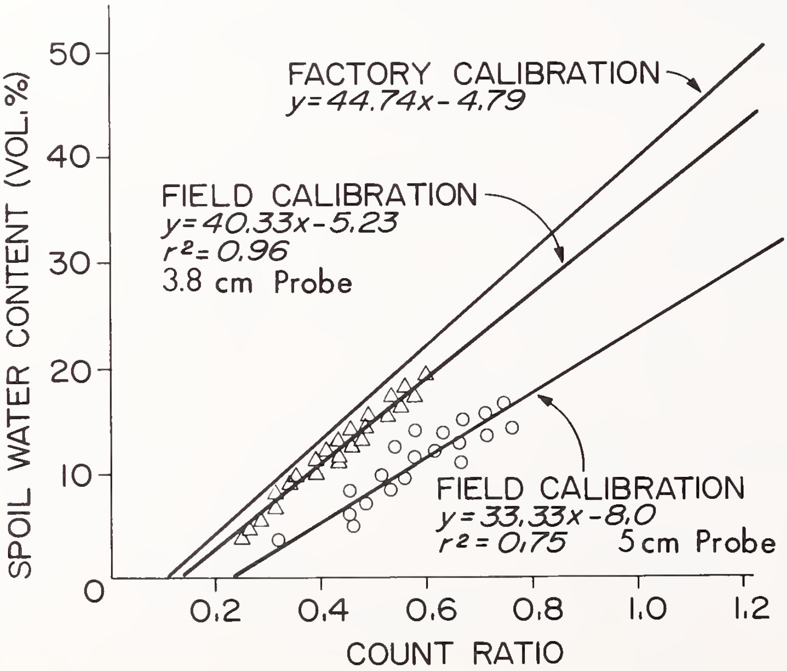


Figure 50. Neutron probe field calibration for two different diameter probes compared to the factory supplied calibration.

Using standard pressure plate apparatus (U.S.D.A., 1969), soil water desorption characteristics at the -0.3 and -15 bar pressure levels were determined for the complete soil profile on samples collected immediately adjacent to each neutron tube. The water content at 0.0 bar pressure for these samples was assumed equivalent to the saturation percentage, determined from saturated pastes (U.S.D.A., 1969).

Bulk densities for each neutron tube site were measured with a Troxler 2601 scaler and 3151 gamma density probe. Factory supplied calibrations were used for the density probe since previous tests indicated acceptable accuracy.

Soil particle size distribution (Day, 1965) was determined for soil profiles located at each neutron tube site. Sand, silt and clay fractions were generalized into particle size categories of 2.0-0.05 mm, 0.05-.004 mm, and <.004 mm, respectively.

Model Clay Cap Test

An area 30 m by 30 m square was leveled in mine spoils and a clay cap 15 m in diameter and 0.5 m thick was constructed over the leveled area. Spoil and clay materials were analyzed for particle size distribution (Day, 1965), clay mineral type by X-ray diffraction, saturation percentage (USDA, 1969), and bulk density using a Troxler surface density gauge with Cs gamma source and Am-Be neutron source. Soil water content within and around the clay cap was monitored with the neutron probe technique. Once the clay cap was in place, 15 aluminum neutron access tubes were installed, each with an aluminum collar at the cap top to prevent water from flowing down the narrow gap between the tube and its access hole (Figure 51). Following access tube

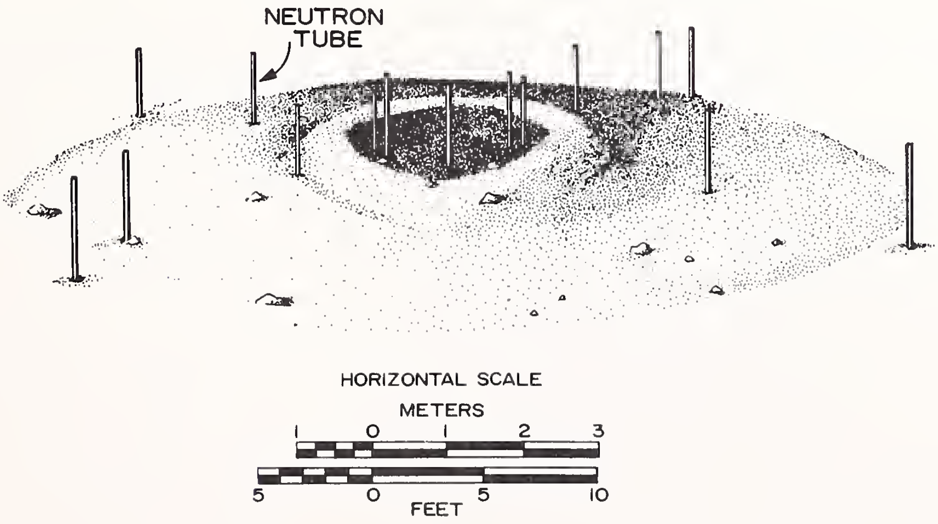


Figure 51. Oblique view of experimental design for model clay cap test.

installation the clay cap was covered with 1.2 m of spoil material. A centrally located earthen dike was constructed to pond water during flooding events.

MODEL CLAY CAP EVALUATION

Physical analyses performed on the scaled down clay cap-spoil system indicated very similar characteristics compared to the large clay cap system constructed during the dragline backfilling operation. The clayey material used to construct the cap contained 42% clay by weight, compared to 49% clay in the larger cap. The mineralogy of the clay in both cap systems was composed primarily of illite and kaolinite, with lesser amounts of smectite, chlorite, and quartz. Both caps were compacted to a bulk density of approximately 1.8 g/cm^3. The saturation percentage (% by volume) of the spoils over the cap was 30%, and that of the clay cap was 40%.

Prior to flooding the clay cap system the spoil water content (volumetric %) around the cap generally ranged from 8 to 16% (Figure 52). The spoil surface was slightly drier due to evaporation, and the clay cap had a somewhat greater water content because of its greater pore space, and corresponding greater water holding capacity. After two days of flooding, 51 cm of water had been applied to the diked area, some spoil material was saturated immediately over the cap. In addition, the wetted front began to mushroom out laterally over the cap. After five days of flooding, 135 cm of water had been applied within the diked area and a portion of spoil material was completely saturated from cap top to the spoil surface. The wetted front continued to mushroom out laterally over the cap, and it was observed that the water content had increased within and immediately beneath the clay cap. During the next 2.5 months the diked area was flooded intermittently resulting in a total of 290 cm of water being applied. Although the clay cap itself had become quite wet and the water content of spoils 0.7 m beneath the cap had increased, the cap continued to be an effective barrier to saturated flow (Figure 52).

Before notable leaching of inhibitory materials from a burial site can occur, water must be present in significant enough quantity to respond to the forces of gravity and flow downward through the spoil profile, i.e. gravitational flow. In comparison, at low soil water contents, leaching of inhibitory material is insignificant since gravitational flow is not present and the resulting flow of water is very small. The clay cap system may permit a small increase in the water content beneath the cap, but the bulk of the water which can cause significant leaching will be effectively diverted away from the zone of inhibitory material.

This clay cap was subjected to atypical conditions. It is infinitely unlikely that the *in situ* dragline backfilled cap-spoil system would ever be subjected to such an intensive flow of water in so short a time. However this heavily irrigated cap clearly demonstrated the principle of water diversion in porous media due to abrupt changes in particle size distribution. Based on this fundamental yet necessary quantitative description of the function of a clay cap, the clay cap concept was tested within an actual surface mine operation.

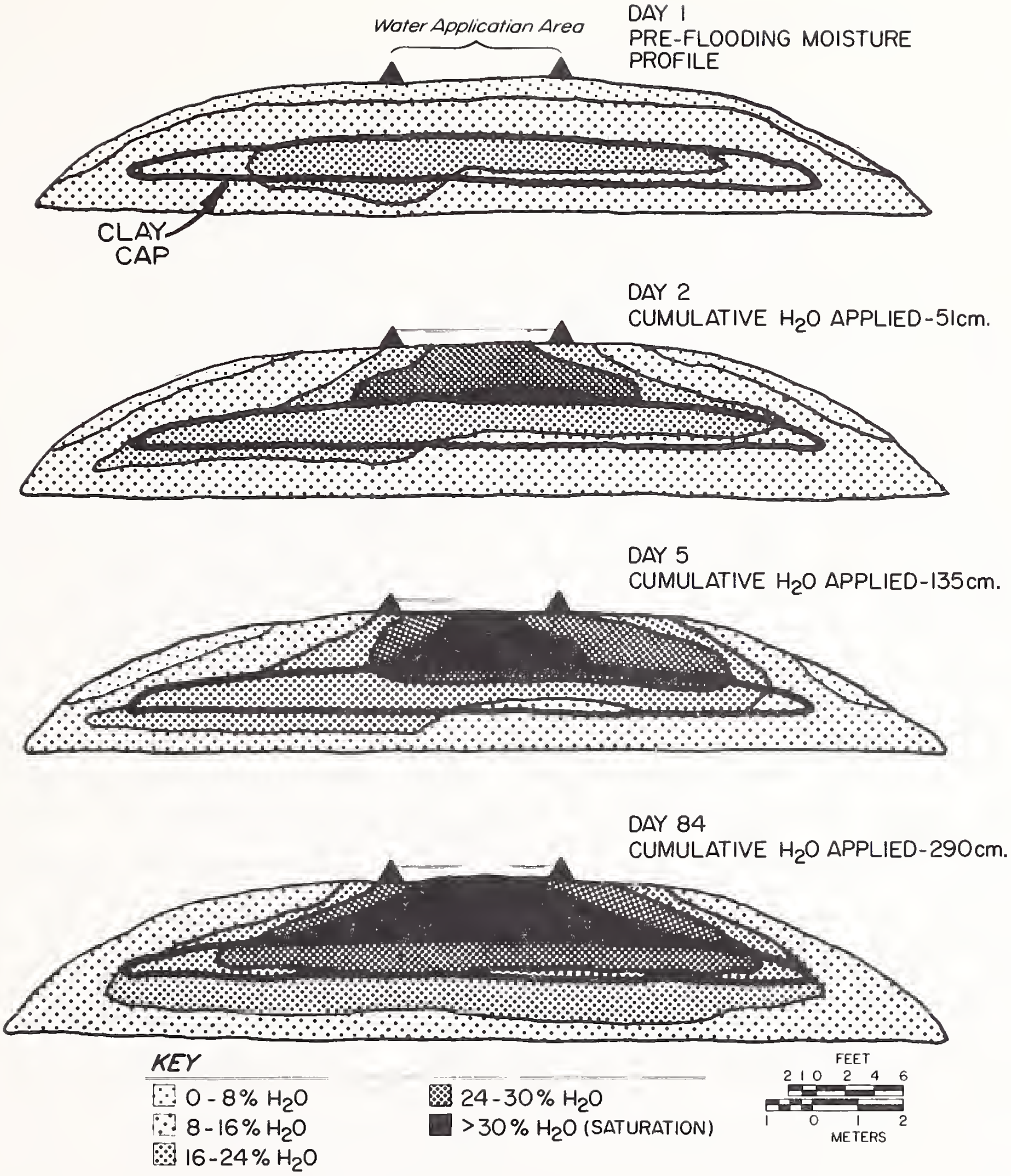


Figure 52. Chronological sequence of water applications to a mine spoil system which contained a compacted clay cap.

HYDROLOGY OF SELECTIVELY PLACED BACKFILL

After final reshaping of the spoil, the selective handling demonstration area was located near a valley bottom in the landscape (Figure 53). The site was nearly level in a NE to SW direction, but tended not to pond water since a 4.3 m (14 ft) elevation difference was present across the 250 m long area. Some runoff water from upslope positions may have accumulated on and infiltrated into the demonstration area spoil material. The demonstration area was sufficiently upslope from the actual valley bottom so that any intermittent surface flows through the main watershed channel would not have intercepted the research site.

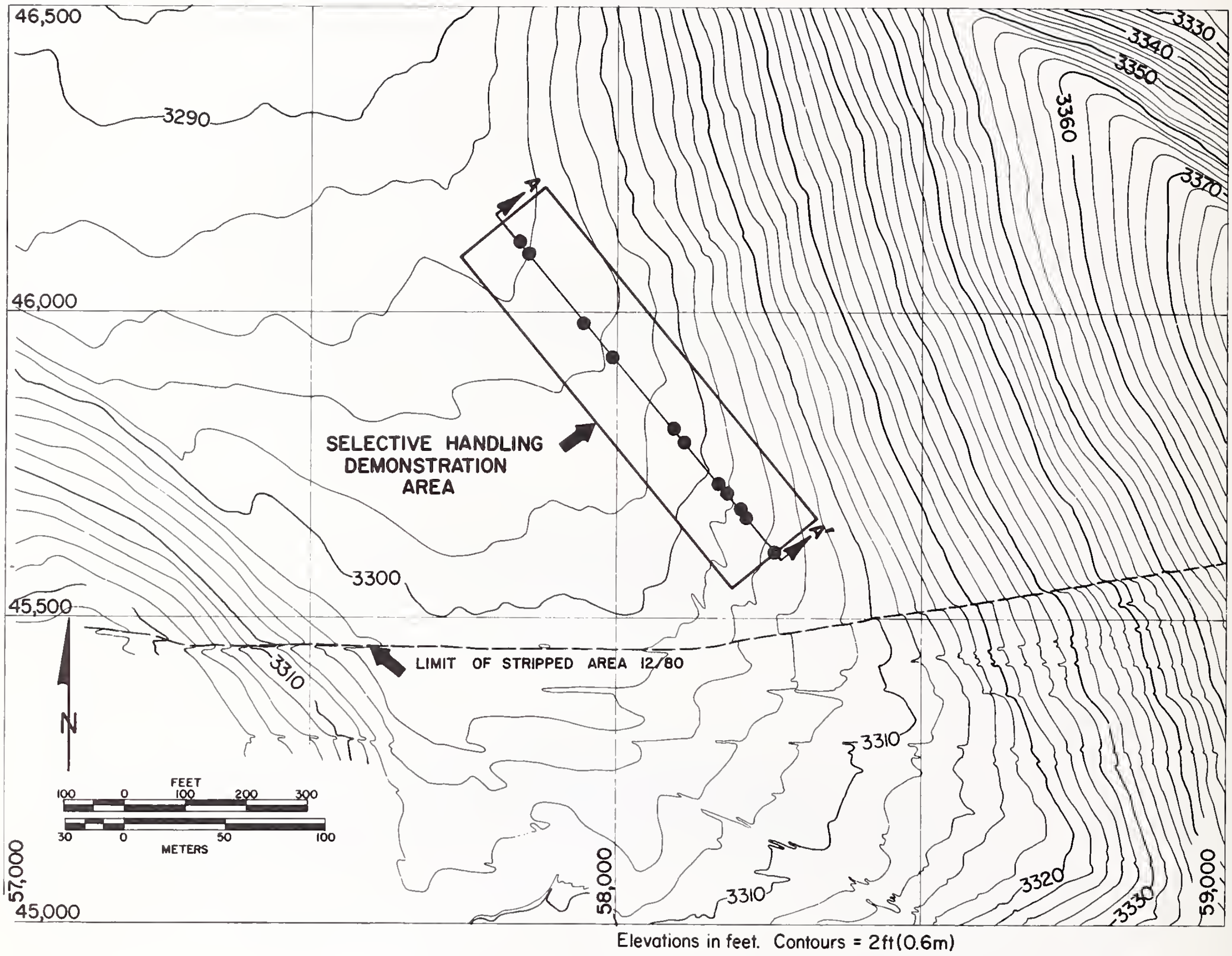


Figure 53. Final mine land surface contours and location of selective handling demonstration at the Western Energy Company Mine Area B near Colstrip, Montana.

Figure 54 shows a mine spoil cross section A-A' which represents the 400 m long backfill area that was constructed in May, 1977. Both saturated and unsaturated spoil-water conditions are shown. Even though spoil profile water contents were determined monthly from November 1977 to December 1980, for purposes of clarity only the wettest and driest profiles are presented for each monitoring site. Other months, if graphed would result in moisture profiles at intermediate positions.

In order to fully understand data presented in Figure 54 this technical note may be required. Soil water content data alone do not indicate the relative wetness or dryness of the soil. For example a soil at 30% water content, could be near saturation (0 bars), field capacity (≃.3 bars), or near the wilting point (≃15 bars) depending on soil texture. To interpret water content data, desorption characteristics of the soil material are presented, which indicate the water content percentage at various soil water potentials, i.e. 0.0 bars, -0.3 bars, and -15 bars. These soil water data

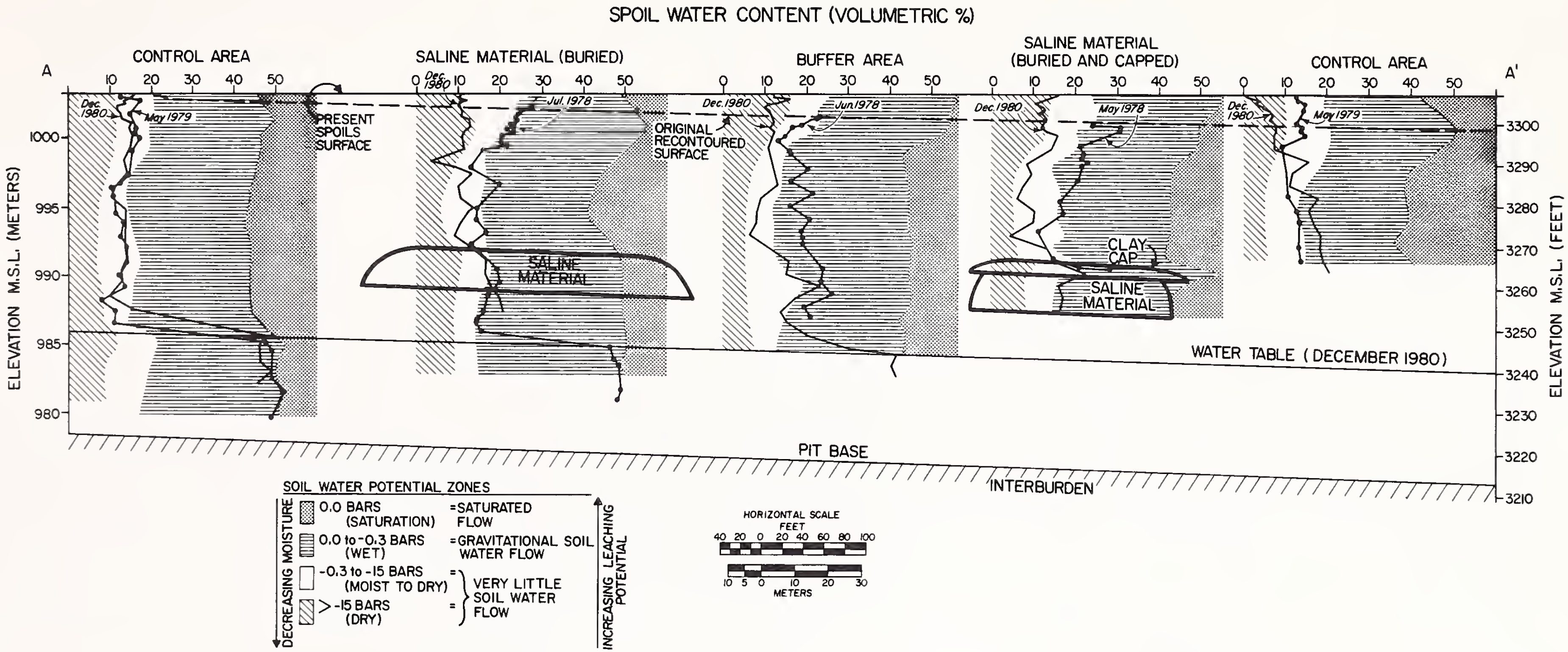


Figure 54. Hydrology of selectively placed backfill material located at the Western Energy Company Mine Area B near Colstrip, Montana.

clearly indicate spoil depths within the backfill area where leaching could take place. For example, the potential for gravitational soil water flow, and thus leaching, is high where soil water content curves consistently remain greater than field capacity (≃0.3 bars), and leaching may become significant as the soil water content curves approach saturation. The approximate ground-water level is apparent where these curves lie near the saturation line. Plant stress due to lack of water is most likely to occur when soil water content curves approach the approximate plant wilting line at 15 bars.

In control areas, located on either side of the buried saline materials, the spoil profile water content ranged between dry (15 bars) and moist (0.3 bars) but never had sufficient water content to permit significant gravitational soil water flow. This was true until the water table was intercepted at the 18 to 20 m depth, then a capillary fringe effect extended approximately 1 m above the water table. The capillary fringe is defined as the layer of soil above the water table into which ground water enters due to capillary rise. In the capillary fringe, ground water is held under tension by the soil and is therefore soil moisture and not ground water, i.e. it is an unsaturated zone. It is important that the capillary fringe not intercept buried inhibitory material since leaching would occur. In order for the capillary fringe to be a factor in the leaching process the water table would have to be within 1 to 2 meters of the buried material (Dollhopf et al., 1979; Gardner, 1965). An effective capillary height of 1 to 2 m is typical for sandy loam material, a texture which dominated this backfill area. Capillary rise can occur to heights >2 m, however at such heights the soil water content within the fringe becomes so small that no significant leaching could occur. The leaching process within a capillary fringe would occur as a function of water table fluctuation. For example, when the water table dropped the capillary fringe would follow and carry dissolved chemical constituents with it. From this basic mechanism, chemical constituents within the capillary fringe could mix into the aquifer zone beneath the water table. Thus it is important that the capillary fringe not come in contact with the buried inhibitory material.

The saline material that was buried, but not covered with a clay cap, lies in a spoil backfill water regime that generally ranged from dry to moist (10 to 25% H_2O or -0.3 to -15 bar zone, Figure 54). These data suggest that during the three year monitoring period a small amount of spoil water moved entirely through the backfill to the water table by unsaturated flow process. Such events occurred during wet spring or early summer periods especially in early July 1978. However, the amount of leaching had to be insignificant since the water regime generally remained in the dry to moist condition, and was not present in sufficient quantity to permit significant gravitational water flow. In December 1980, the water table was present approximately 4 m below the buried saline material and a 1 m effective capillary fringe was generally detected in this zone.

Some soil water moved by unsaturated flow processes from the surface to the clay cap during wet spring periods, especially during May 1978. Although the spoil material immediately above and below the cap and the cap itself were not near saturation, the cap apparently impeded water flow from above, which resulted in a marked increase of water content in this zone. For example, the water content of the cap during construction in May 1977 was approximately 15%, and during the following three year monitoring period the cap moisture content was as high as 29%. At 29% moisture content (0.0 to -0.3 bar zone) in the clay cap, some downward water flow by gravitational forces occurred during April to July 1978 period, but the amount was small. The saline material beneath the cap remained in a moist condition, meaning no significant leaching occurred.

SUMMARY

A 26 m thick backfill area, where saline materials were selectively buried, was monitored for a 3 year period regarding aquifer establishment and unsaturated flow from surface to the pit base. The burial site was located near the foot of a slope, but not in the valley bottom of a mined landscape. The saline material remained 4 m above the reestablished water table which was 8 m above the bottom of the pit. Likewise the buried saline material was 11.5 m below the surface, thereby preventing interaction with the future root zone.

Unsaturated water movement events occurred from the surface to the buried saline material during wet spring periods. However, the quantity of flow was not substantial and no significant leaching of salts from the buried saline material, either clay capped or not capped, was expected.

The intent of the clay cap over buried saline material was to divert water flows so that salt leaching into the deeper aquifer would be prevented. The soil water regime associated with the clay capped saline material was not different from the uncapped saline material. During the three year monitoring period both wet (1978) and dry (1980) spring precipitation periods occurred at the site (see Figure 3). Therefore, construction of clay caps over buried material in the semiarid western states may not be necessary. This is true when the burial site is positioned beneath a hilltop or slope in the mined landscape. If the burial site has to be constructed in a valley bottom, or other land morphological feature that would collect water, then a clay cap may be advisable.

VIII

SPOIL MIXING PHENOMENA

Selective burial of overburden inhibitory zones in order to hydrologically isolate such materials may cost 1.1 to 1.5 times more than the normal spoiling operation, as discussed in Chapter VI. Before such methods are practiced it may be well to quantify whether normal dragline spoiling can sufficiently mix problem overburden material to produce an environmentally acceptable spoil profile.

Prior to strip mining, overburden is composed of a more or less orderly array of approximately parallel rock strata. These strata may contain materials which exceed guideline levels considered potentially harmful or inhibitory to plant growth. The rock strata of concern underlie the topsoil, and are for the most part below the biologically active surface zone. However, after the overburden has been spoiled these materials may come nearer to, or intrude into the biologically active area, at which time the concentration of soluble salts and heavy metals may become of concern. The effect these spoils will have upon the water quality of the reestablished aquifer is also of concern, especially where the original and newly reestablished aquifers are important resources. This concern stems from the possibility that ground water will take on, to a certain extent, the chemical characteristics of its flow medium.

There is a tendency for the overburden stratigraphic sequence to be inverted when transferred into spoil piles, however such strata are not transferred as whole units. Rather, several strata may be crossed by the dragline bucket as it is filled and successive loads are collected from varying depths. The greatest degree of mixing occurs as the bucket is unloaded and its contents spread both horizontally and vertically over a larger area than originally occupied. This digging scheme and spoiling of the overburden tends to mix the original stratigraphic conditions, and may result in spoils of increased chemical and physical homogeneity when compared to the original condition. As a result, zones or strata of potential toxicity may be reduced to acceptable levels which are neither harmful to plant growth nor to water quality.

The following discussion describes an experiment designed to document the mixing phenomenon and the degree of homogeneity achieved using three spoiling regimes; scatter spoiling, normal spoiling, and dump spoiling.

METHODS

Experimental Design

The demonstration site was located in mine pass N-7 and the experimental design consisted of three treatments; scatter spoiling, normal spoiling and dump spoiling (Figure 55). The mine pass was approximately 60 m wide and about a 60 m length of the pass was used to test each spoiling treatment. Prior to spoiling the overburden, the demonstration area was sampled with two transects (A-A', B-B') comprised of drill holes on a 30 m grid (Figure 55). Composite samples representing 1.5 m depth intervals were collected from the overburden surface to the coal surface from each drill hole. Each sample was analyzed

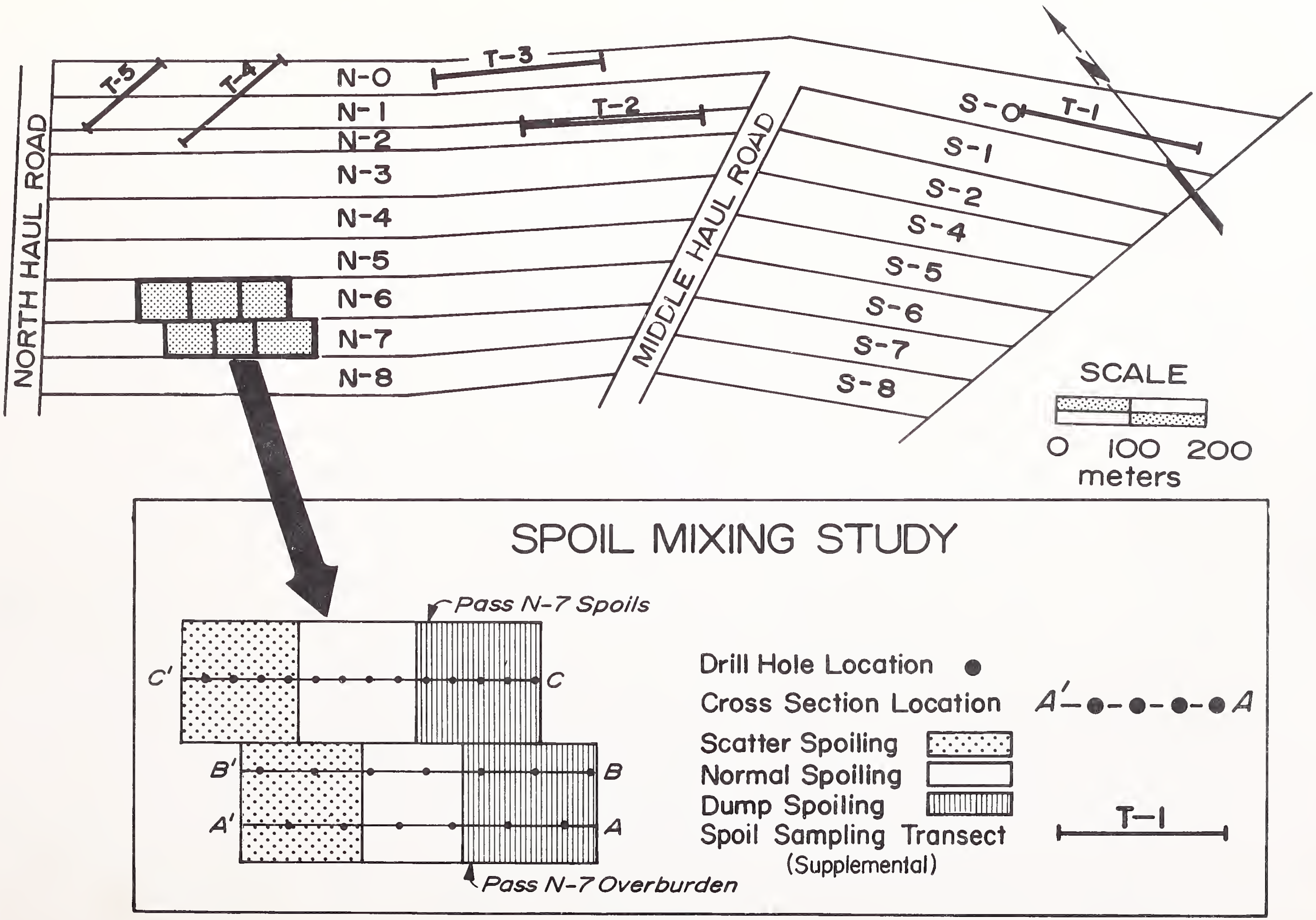


Figure 55. Overview of mine passes with the location and experimental design of the spoil mixing study, Western Energy Company Mine Area B, near Colstrip, Montana.

for electrical conductance (EC), zinc (Zn), nickel (Ni), lead (Pb), and percent sand, silt and clay (Appendix C).

Progression of the Marion 8050 dragline, which had a 46.2 m^3 (60 yd^3) bucket and 99 m (325 ft) boom and which allowed a versatile working radius of 87.2 m (286 ft), was in a northerly direction and pass N-7 overburden was spoiled into the pass N-6 pit (Figure 55). The demonstration began with scatter spoiling, followed by normal spoiling, and finally dump spoiling. The normal spoiling treatment served as the control and represented the usual routine of the dragline operation.

When the overburden of interest was spoiled by the dragline, a road was constructed on the material to permit access of the drill rig. The spoil material was sampled with one transect (C-C', Figure 55) comprised of 13 drill holes at 15 m intervals. Composite samples representing 0.9 m depth intervals were collected from the spoil surface to the base of the pit. Each sample was analyzed in an identical manner as the overburden.

Dry rotary drilling with compressed air was the method used for all sample holes in overburden and spoil material. By comparing the pre-spoiled overburden chemical and physical cross-sectional framework to the post-spoiling cross section the magnitude of the mixing was ascertained for each treatment.

Spoiling Description: Scatter, Normal, Dump

The dragline bucket excavates an overburden slice approximately 1 m deep and 15 m long. Normal dragline spoiling generally results in the bucket load of material being cast over an area 20 m long by 10 m wide. This basic technique was quite constant between dragline operators, although their mannerisms and styles differed. When initiating spoiling into an adjacent pit the normal process was to initially dump spoil, that is, a bucket load was dropped when it was nearly stationary. The dumped material acts as a base and assures a base without entrapped water, thus the spoil pile will be less likely to slump or settle. Once this initial fill was in place, the normal spoiling method was used for the bulk of the overburden. A casting technique was used at times when a maximum reach was needed, such as when placement of a load on the backside of the spoils was desirable.

Scatter spoiling, as defined here, was basically an exaggeration of the normal method. Its uniqueness was the degree each bucket load was spread over the spoil pile, which was generally 40 to 60 m long by 10 m wide (Figure 56). In order to spread the overburden this amount, the dragline would continue its swing while dispensing the load. The operator was able to control the rate of the load leaving the bucket, hence the amount of spreading.

The term "dump spoiling" was used to describe the process whereby the bucket of material comes to a complete halt in its swing before the load was dropped (Figure 56). In this demonstration each successive bucket load was dropped approximately over the same point. The spoils formed in this manner were cone shaped. Some of the spoils from any given drop would remain on the peak, increasing its height, the remainder would slide down the pile in varying amounts.

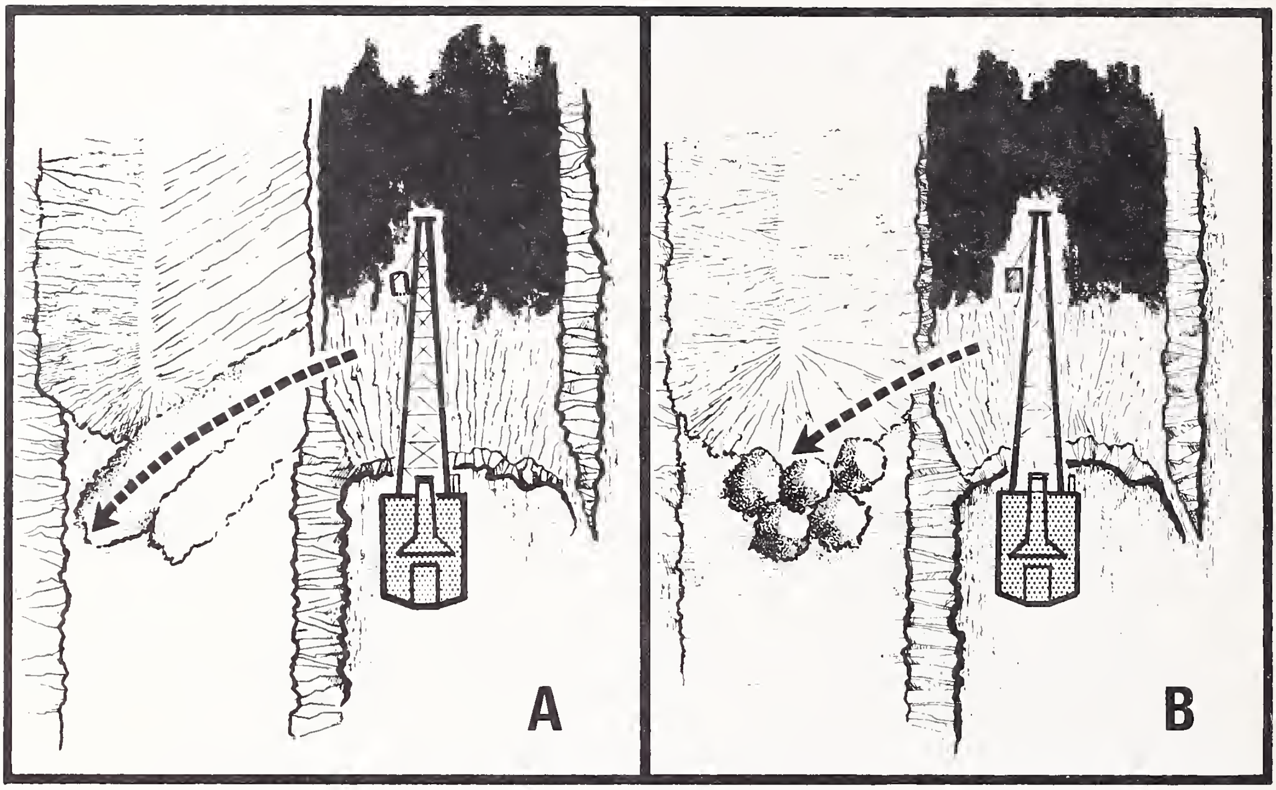


Figure 56. Aerial view of dragline performing scatter spoiling (A) and dump spoiling (B).

To summarize, each filling of the dragline bucket resulted in an overburden slice approximately 1.2 m deep and 30 m long. During scatter spoiling the operator would drop this material over a distance of as much as 60 m. During normal spoiling this material would be dropped over a distance of about 20 m, and dump spoiling resulted in material being dropped over about a 10 m distance. It is difficult to hypothesize whether one spoiling method will have an advantage over the other, or whether spoiling in general can dilute problem overburden zones to an environmentally acceptable result since there are many variables. The following study may help quantify these unknowns.

RESULTS

Field Operation

It was expected that the 180 m long experimental area would require four days of spoiling work, however, a 30 day period during December 1977 and January 1978 was required due to dragline operational problems. These problems were not attributable to the spoiling techniques being tested, rather the major delay was caused by the breakdown of a swing motor.

No economic analysis will be presented comparing the dragline efficiency of operation as a function of these three spoiling methods. However, general observations will be briefly discussed. The normal spoiling operation appeared to be the most efficient manner in which to unload the bucket, because release of the load, deceleration and return swing could be smoothly incorporated into a single event. The dump spoiling technique probably decreased the dragline stripping rate (m^3/hr) since the dragline would come to a complete stop prior to dumping. The hesitation required by this action of the machine would have the effect of decreasing the stripping rate for this spoiling method. Scatter spoiling may have also decreased the dragline stripping rate slightly since the swings were longer than normal and somewhat more difficult to perform.

Theoretical Mixing

A useful process would be to mix overburden material that is considered chemically or physically unsatisfactory for reclamation with material that is desirable for purposes of reclamation. For example, a surficial overburden zone with a high concentration of lead, e.g., 15 ppm may be mixed during the dragline spoiling process with material at a lead concentration of 1 ppm, resulting in a spoil pile lead concentration ranging between 3 and 5 ppm. This resultant spoil material if left near the surface would not be harmful to plant growth, or if left at the pit base where an aquifer may develop it would tend to give such water a characteristically lower lead concentration. This principle would similarly apply to overburden physical parameters such as sand and clay content.

Ideally the principle of overburden mixing should be a linear function. However, overburden is a complex medium and properties such as texture, clay mineral, and pH could affect the relationship of the spoil mixture.

Therefore, utilizing spoil material from the research area, the principle of overburden mixing was demonstrated in the laboratory for the parameters, lead (Pb), zinc (Zn), nickel (Ni), electrical conductance (EC), sand, and clay. With but a few exceptions the relationships were highly linear, the slope of which varied as a function of the concentration present in materials mixed (Figure 57). For example, a sample with 9% clay content was mixed with a sample at 57% clay in proportions of 75/25, 50/50, and 25/75, resulting in samples of 25%, 40%, and 49% clay content, respectively; a near linear relationship. Mixtures regarding trace elements and soluble salts behaved in a similar manner.

These results indicate that potentially inhibitory overburden materials could be made acceptable through mixing. The degree of mixing obtainable will be a function of both the range in concentration of materials mixed and the mixing efficiency of the dragline spoiling technique. The ability of a dragline to mix overburden materials will be addressed in the following section.

Dragline Mixing

As described in the methodology, cross-sectional distributions of chemical and physical properties were prepared for both the overburden and resultant spoil material (Figures 58-62). The top two cross sections (transects A-A', B-B') represent overburden while the bottom cross section (transect C-C') represents the dragline spoiling results as a function of scatter, normal, and dump spoiling techniques. An aerial view location of drill hole transects A through C is shown in Figure 55.

All the cross sections indicate a substantial volume of fill material within the dump spoiling region of the overburden, which requies a note of

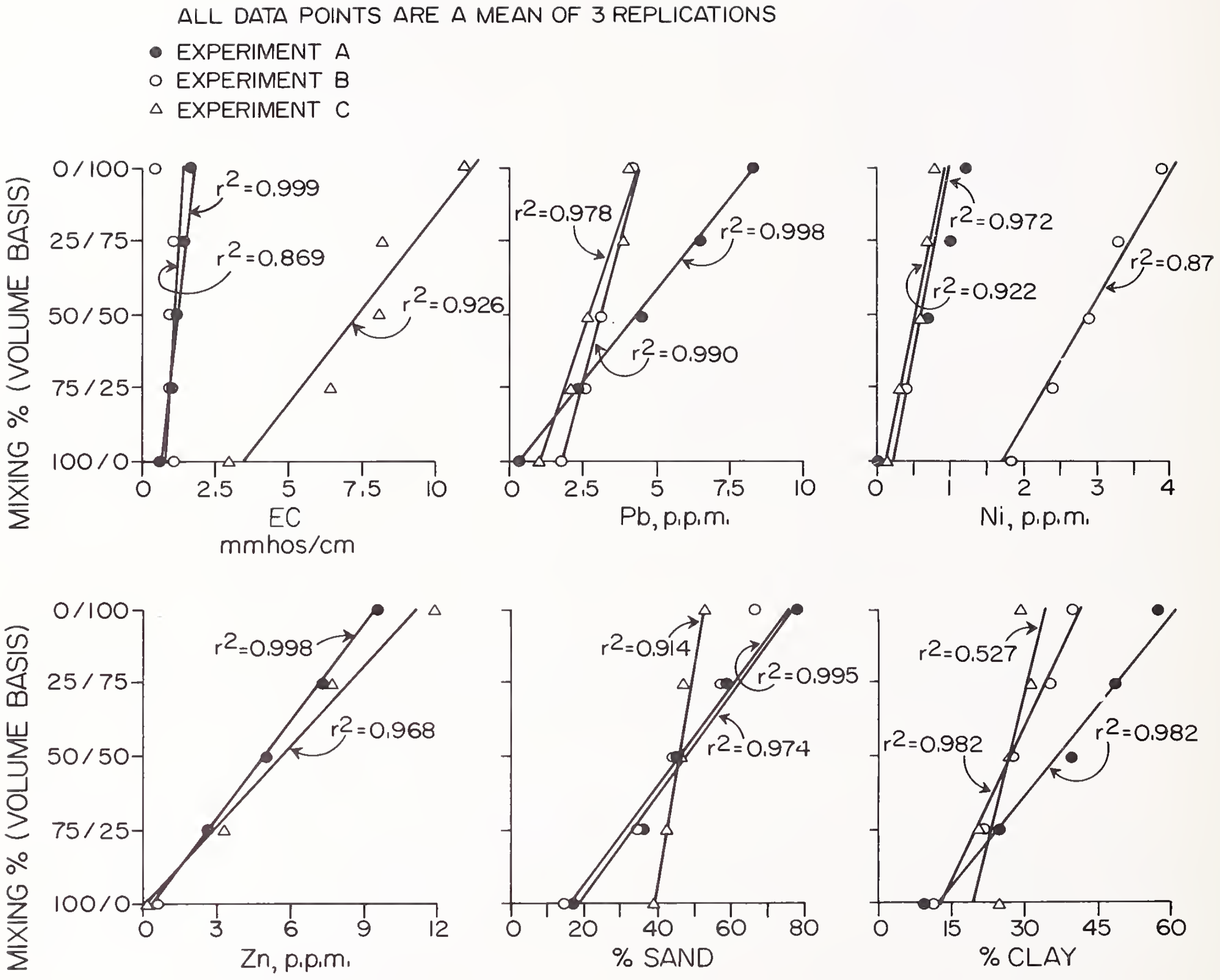


Figure 57. Relationship between chemical or physical dilution in mine spoils and degree of mixing. Regression coefficient (r^2) values near 1.0 indicate excellent correlation of data.

explanation. A problem was caused by overburden unable to support the dragline mass which resulted in differential settling of the body sufficently deep that the machine could not move under its own power. To rectify this problem, overburden under the machine was removed to an approximate depth of 7.5 m. The void created was filled with clinker (porcelanite) to form a solid foundation for the machine to set upon. The immediate vicinity of the dragline was also prepared in such a manner to facilitate the walking movement of the dragline. Overburden removed from beneath the dragline was temporarily placed on the adjacent highwall for later rehandling. Therefore a substantial volume of clinker fill was introduced into the dump spoiling portion of the experiment, and a part of the overburden was rehandled.

The overburden consisted of a large stratum of inhibitory level (>70%) sand, which occupied about half of the overburden and was enveloped, both above and below, by less sandy strata (Figure 58). After spoiling, the integrity of the sandy zone was disrupted throughout the three treatment

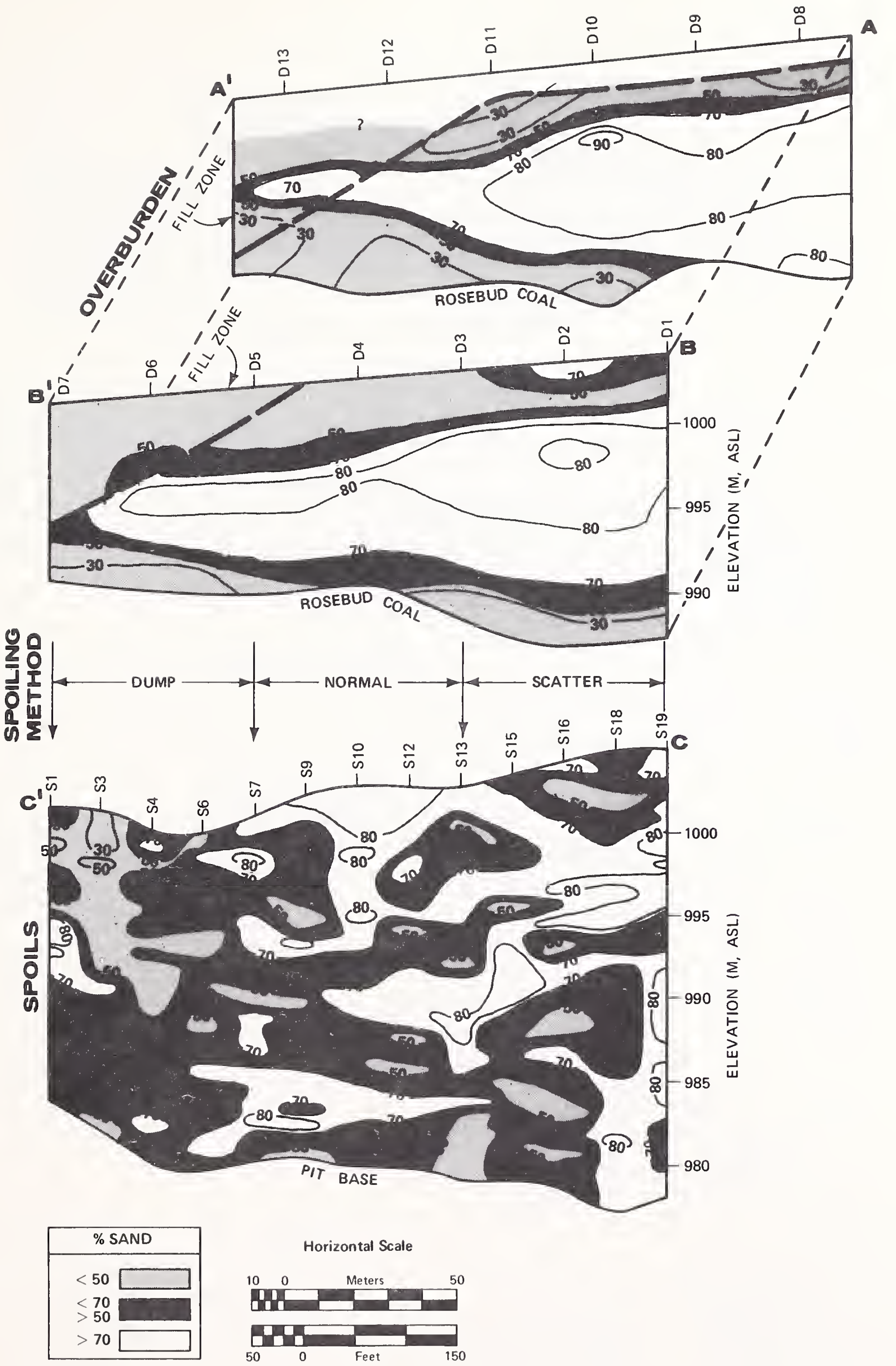


Figure 58. Cross-sectional distribution of sand content in overburden (A-A', B-B') and resultant spoils (C-C') demonstraing mixing phenomena for different dragline spoiling techniques at Colstrip, 1978. See Figure 55 for borehole locations.

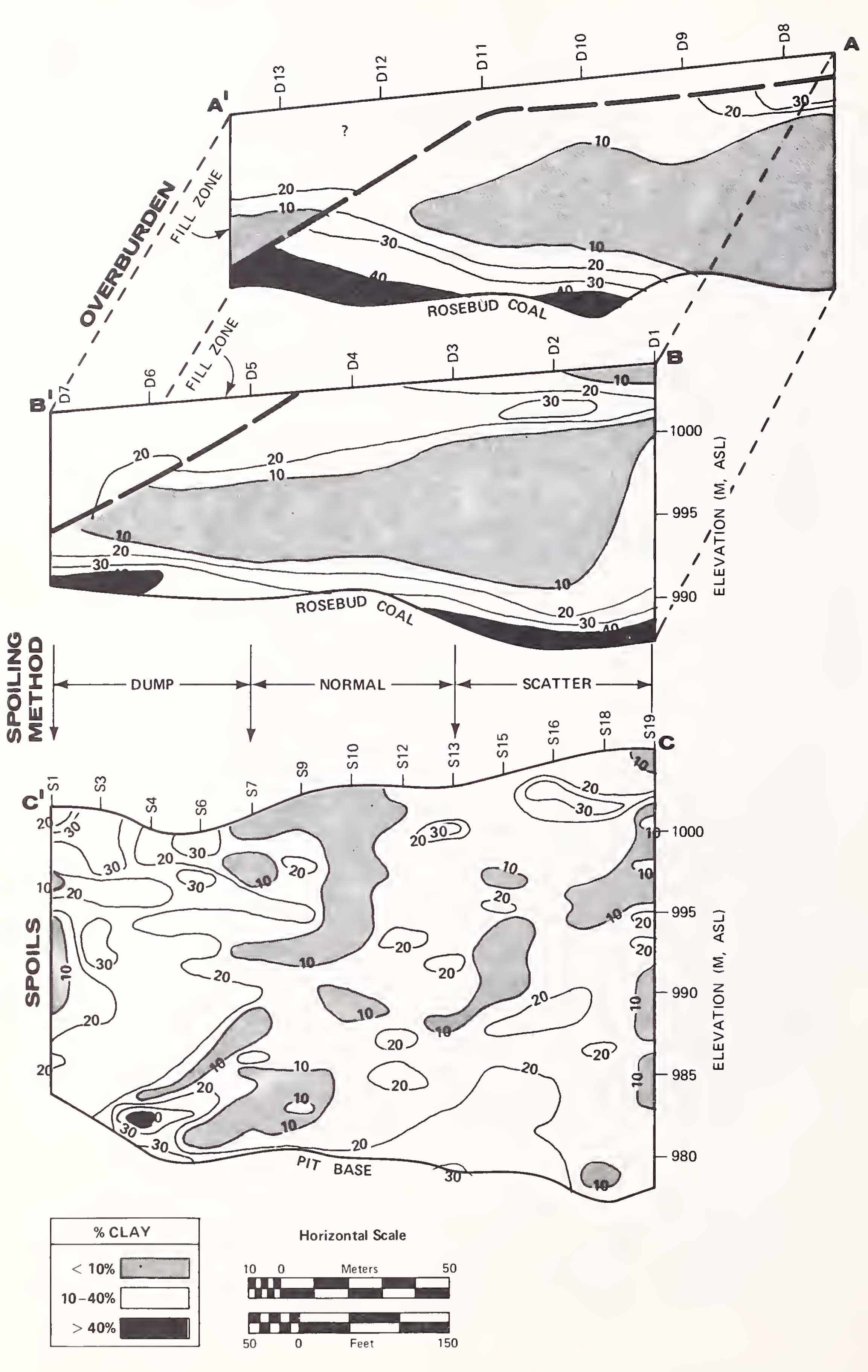


Figure 59. Cross-sectional distribution of clay content in overburden (A-A', B-B') and resultant spoils (C-C') demonstrating mixing phenomena for different dragline spoiling techniques at Colstrip, 1978. See Figure 55 for borehole locations.

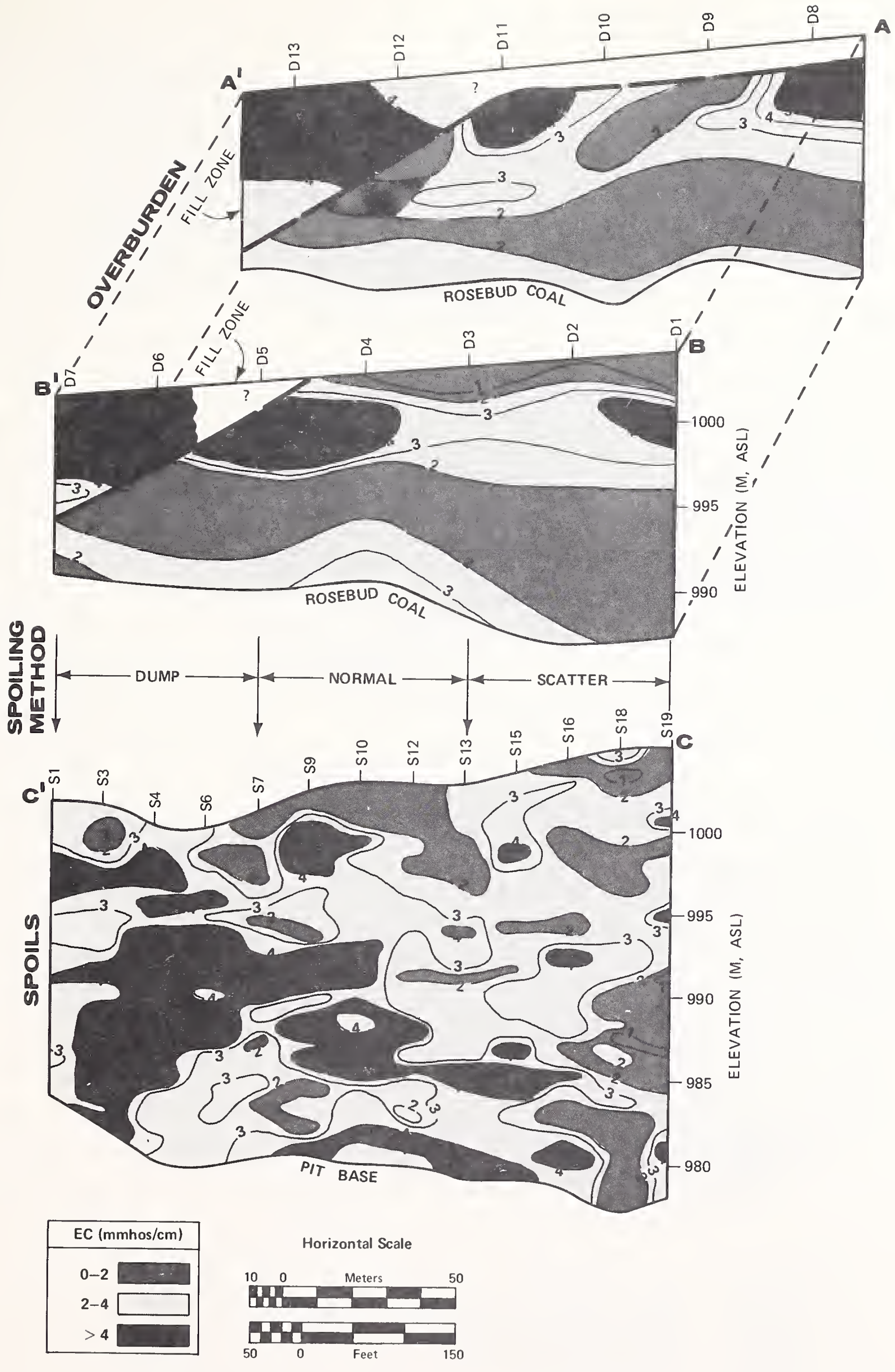


Figure 60. Cross-sectional distribution of electrical conductance (EC) content in overburden (A-A', B-B') and resultant spoils (C-C') demonstrating mixing phenomena for different dragline spoiling techniques at Colstrip, 1978. See Figure 55 for borehole locations.

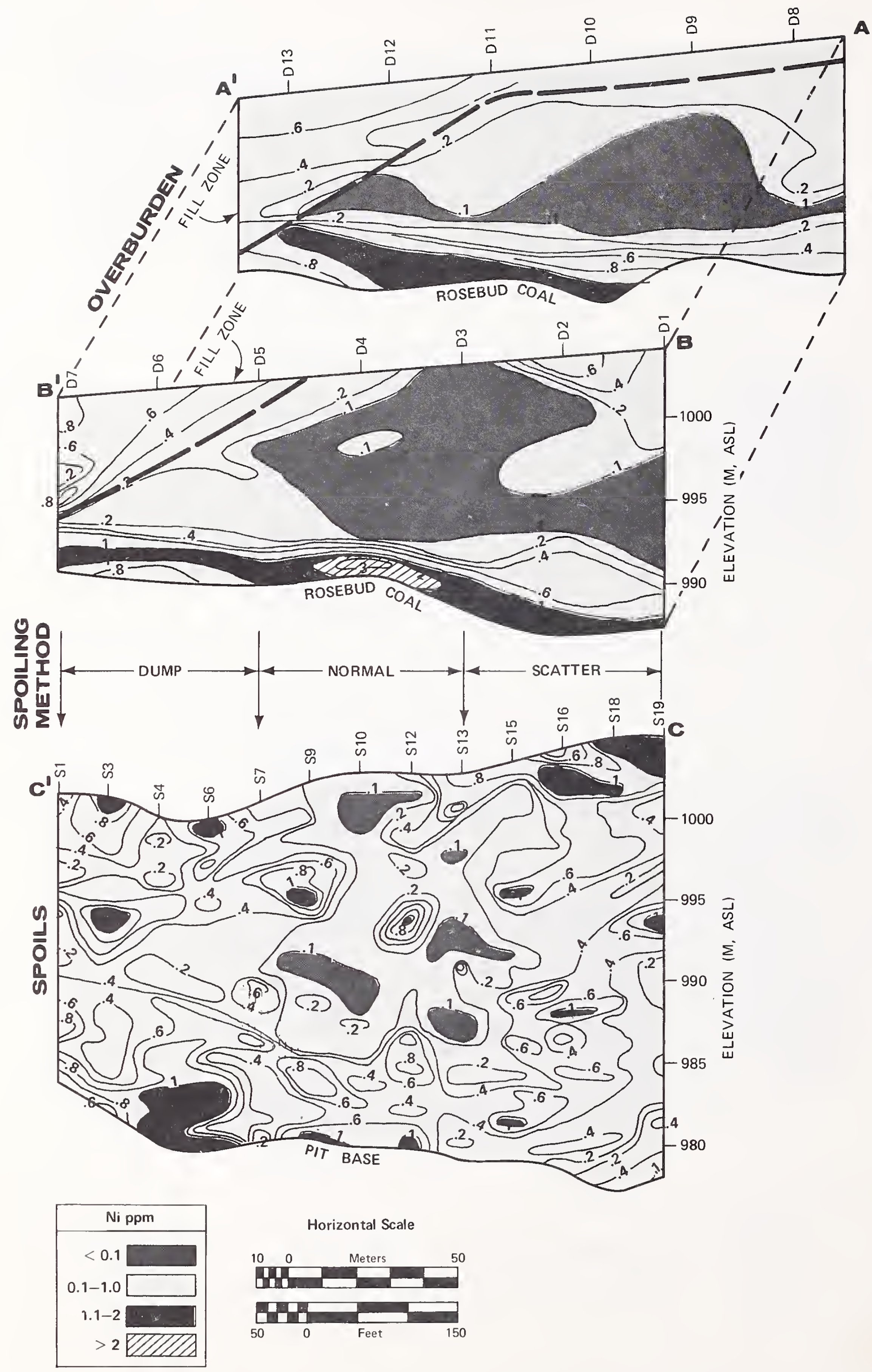


Figure 61. Cross-sectional distribution of nickel (Ni) content in overburden (A-A', B-B') and resultant spoils (C-C') demonstrating mixing phenomena for different dragline spoiling techniques at Colstrip, 1978. See Figure 55 for borehole locations.

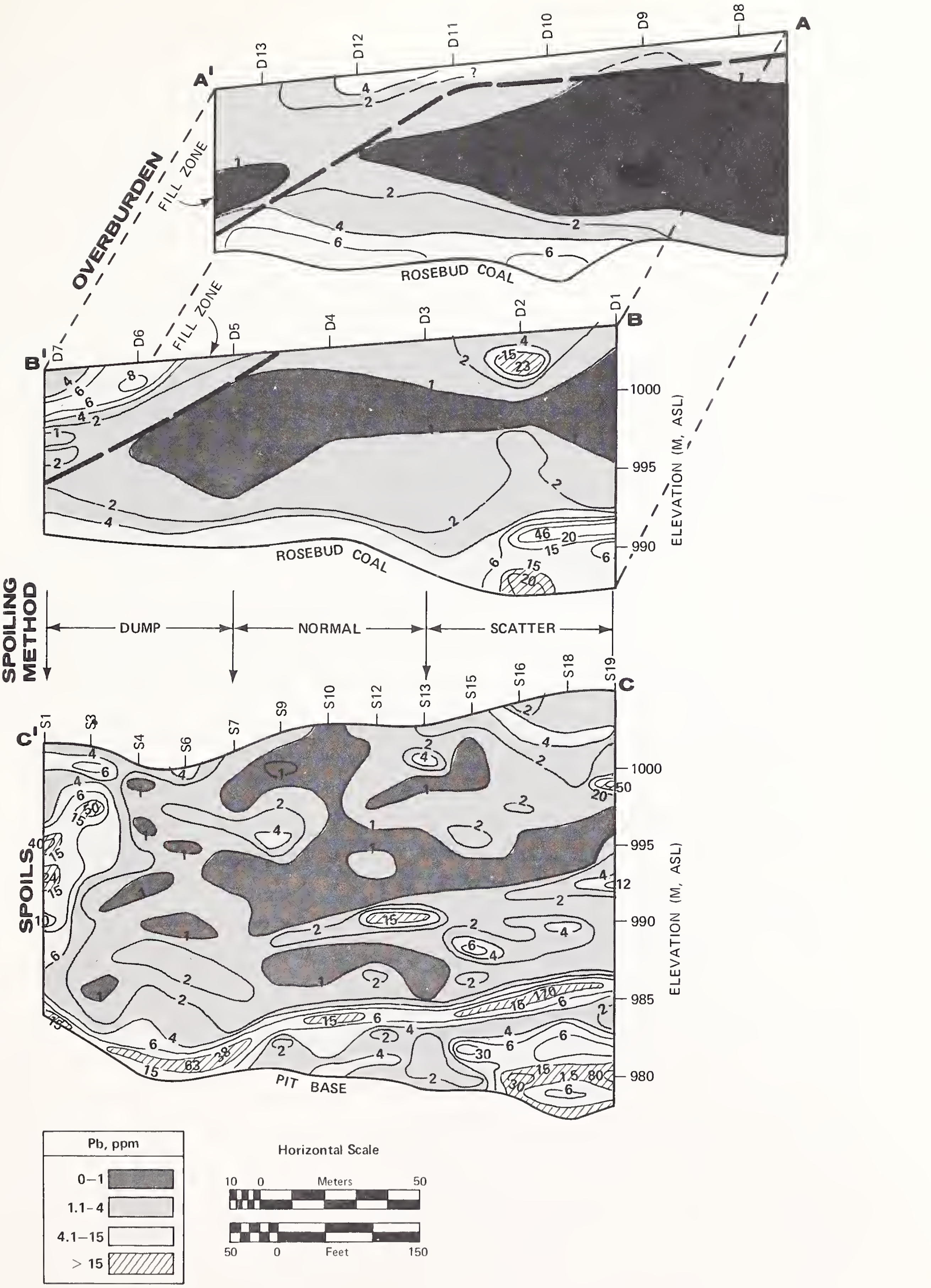


Figure 62. Cross-sectional distribution of lead (Pb) content in overburden (A-A', B-B') and resultant spoils (C-C') demonstrating mixing phenomena for different dragline spoiling techniques at Colstrip, 1978. See Figure 55 for borehole locations.

areas. The overburden which was scatter spoiled contained, volumetrically, at least half inhibitory level sand. Spoiling seemed to result in some reduction of the high and low sand regions; indicating that mixing of these extremes occurred. The area which was spoiled in a normal manner showed similar trends. However, in both scatter and normal spoiled areas, inhibitory sand material remained within the spoil pile. Apparently, the dilution potential of these spoiling techniques was exceeded when half of the overburden was composed of sandy (>70%) material.

It is worthy to note that inhibitory sand material was found largely below the root zone in overburden, but the normal and scatter spoiling processes brought much of this material to the surface. Such a result presents an argument favoring selective handling of overburden in order to keep such sandy materials out of the root zone. Soils with greater than 70% sand may not retain desirable levels of water for plant production. The dump spoiled overburden seemed to result in relatively little mixing.

Inhibitory clay strata (>40% clay) were located adjacent to the coal in thin bands of overburden (Figure 59). All spoiling treatments were effective in diluting these clay strata to acceptable levels in the spoil pile.

The presence of an inhibitory clay pocket in the dump spoiled area, the only such pocket in the spoil ridge investigated, suggests that this technique may be less effective in the dissipation and dilution of inhibitory clay deposits. It is interesting to note that although the clay strata were adjacent to the coal in overburden, such strata did not reappear near the surface of the spoil ridge. That this result might occur stems from the belief that materials near the pit base are spoiled towards the top of the spoil ridge. Apparently, that is not necessarily true and normal spoiling can sufficiently dilute 2 m thick clay zones adjacent to the coal when much of the 14 m of overburden, as in this case, contained >70% sand.

Examination of electrical conductance distribution in the overburden indicated the presence of potential problem saline zones (>4 mmhos/cm) near the surface, while the majority of the overburden was found acceptable (Figure 60). Overburden thickness was about 14 m and the saline zones, which were not horizontally continuous, were about 3 m thick.

The spoil profile (Figure 60) indicated the scatter spoiling operation may have been somewhat more effective in diluting salt affected overburden zones than either the normal or dump spoiling techniques. The pre- and post-scatter spoiled materials suggested much of the overburden salt affected zone was successfully diluted, however an amount still remained in the spoil pile as many smaller fragmented zones, distributed from near surface to the pit base. The normal and dump spoiled areas also demonstrated fragmentation of the overburden salt zone, but in terms of volume there did not appear to be any substantial reduction in salt affected material. The potential for dilution of the 3 m thick salt affected zone in the overburden assigned to be scatter spoiled appeared to be greater than that of the other two treatment areas since more overburden material of a lower salt content was available for mixing.

A rather continuous 2 to 3 m thick undesirable nickel (>1.0 ppm) zone was present in overburden adjacent to the coal seam (Figure 61). Spoiling by the three methods resulted in this high Ni level stratum being largely diluted. However the integrity of some of the stratum was retained as fragmented zones found throughout the spoil pile. Some of the Ni strata from the base of the overburden was found near the surface of the spoil pile, conceivably in the future root zone. In this particular example, final grading of the spoil piles may bury the potentially inhibitory Ni material below the root zone.

Regarding Ni dilution, there seemed to be little difference between spoiling techniques. The normal spoiled region resulted in the least amount of spoil material with Ni in excess of 1.0 ppm, indicating a better degree of dilution.

An important point can be made here. The location of strata for inhibitory levels of nickel and clay were in close association (Figures 59, 61). However the nickel apparently did not dilute nearly as well as the clay during the spoiling process. This indicates that success in diluting inhibitory strata will be dependent on more than just efficient dragline mixing techniques, suggesting a substantial range in concentration of strata may be a necessary prerequisite. Apparently, the large range of clay content in the overburden, i.e., much of the overburden contained >70% sand, favored nearly complete dilution of the clay strata, while the concentration gradient for Ni may not have been as favorable.

Use of Pb and Zn data for the evaluation of the described spoiling treatments are not reliable for this experiment because of a definite contamination problem. Peak Pb and Zn concentrations were observed in the spoils precisely where drill stems were changed during spoil sample collection.

When an additional drilling stem was attached (at 6 m intervals), a Pb based thread joint compound was applied. Depending upon the amount used by the driller's helper and other variables, the amount and duration of the contamination would vary. Consequently Pb levels in spoil samples were drastically affected. As clearly shown in Figure 62, a consistently high Pb layer was indicated at the 18 m depth which corresponds to the depth where an additional section of drilling stem was added. Peak concentrations were also present at 6, 12, and 24 m depth intervals.

The spoil profiles for Zn were likewise influenced by the joint compound, although to a lesser degree. Both Pb and Zn concentrations in excess of the original overburden were indicated to be present in the spoils as a result of the contamination.

These Pb and Zn data have been significantly affected by drill stem lubricant contamination necessitating their deletion from the evaluation of the three spoiling treatments. Apparently the joint compound contained little of no Ni, for the potentially inhibitory zones of Ni did not consistently appear at depths corresponding to drill stem additions. We included Figure 62 in this discussion since it clearly demonstrates the problem and indicates such joint compounds should be eliminated from overburden investigations on trace elements.

To aid in interpretation of these data, statistical means and standard deviations were determined on the various parameters analyzed in overburden and spoil material for each spoiling method (Table 21). The low standard deviation for each chemical and physical parameter in spoils compared to overburden suggests a more homogenous material. This result would be expected due to the mixing action of the spoiling process.

Table 21. Mean ($\bar{x}$) and standard deviation (SD) of chemical and physical properties in the overburden mixing study.

	% Sand $\bar{x}$ ± SD	% Clay $\bar{x}$ ± SD	EC mmhos/cm $\bar{x}$ ± SD	Ni(ppm) $\bar{x}$ ± SD
	Scatter Spoiling			
Overburden	67.3 ± 21.6	16.6 ± 15.4	2.5 ± 1.5	0.24 ± 0.39
Spoil	67.1 ± 13.4	16.1 ± 6.8	2.7 ± 1.1	0.47 ± 0.36
	Normal Spoiling			
Overburden	62.5 ± 21.2	17.5 ± 11.8	3.5 ± 2.6	0.34 ± 0.64
Spoil	67.4 ± 11.7	13.7 ± 5.8	3.3 ± 1.4	0.37 ± 0.32
	Dump Spoiling			
Overburden	46.9 ± 25.2	27.8 ± 15.6	3.7 ± 2.2	0.61 ± 0.52
Spoil	52.6 ± 13.7	21.8 ± 8.9	4.2 ± 2.2	0.58 ± 0.40

SUPPLEMENTAL *IN SITU* MIXING STUDIES

The objective here was to return after grading and recontouring to the location of the spoiled inhibitory zones to relocate, if present, those areas of potential inhibitory spoils. Such information would contribute to the understanding of how successfully inhibitory overburden strata were diluted during the normal spoil process.

Overburden samples were collected on a 60 m grid intensity concurrent with the stripping operation of mine passes 0, 1, 2 and 3 at the Western Energy Company Mine Area B (Figure 55). Analysis of these samples indicated areas of inimical clay, EC, and Ni in segments of the various mine passes. The areas affected by one or more parameters represented from < 10% to nearly 100 % of the overburden mass.

Sample site locations were determined by examination of the pre-spoiled overburden data for strata of inhibitory material. Five such locations within the mine area overburden contained varying volumes of inhibitory material, each with unique characteristics of distribution. Using aerial photos and ground survey the resultant spoils from these overburden zones were located. Transects T-1 through T-5 (Figure 55) were established on the regraded spoils where the greatest likelihood of locating the inhibitory materials existed. Drill sites were established at 15 m intervals along each transect. Samples were collected in 1.5 m increments as air ejected cuttings to the Rosebud coal in overburden and to the pit base in spoils. Analysis procedures were outlined earlier in this chapter.

Spoil transect T1 was located at the south end of the mined area (Figure 55). Inhibitory levels in the overburden for clay, EC and Ni had been located. Electrical conductivity and Ni affected areas constituted 10% or less of the original overburden which had a depth of 30 to 50 m. Twenty five to 50% of the overburden mass contained excessive levels of clay.

Sample analyses from transect T1 indicated the presence of pockets of spoils with potentially inhibitory EC and Ni levels; although a smaller volume of spoil material showed higher EC and Ni levels than in overburden. The rather large volumes of clayey material in overburden were partially diluted to acceptable levels in the spoil pile, but considerable volumes of clayey material (>40%) remained.

Transect T2 was located northwest of T1 on spoils from mine pass 3 (Figure 55). The overburden which resulted in spoils where transect T2 was located contained discontinuous zones of high clay and Ni, making up less than 10% of the overburden mass, also a large saline zone was present in the upper 5 m of overburden which was 20 m thick.

The volume of saline material in the spoils was substantially reduced in the spoiling process (≈10% of the spoil mass). Pockets of saline material were observed at or near the surface and bottom of the spoils. No inhibitory clay concentrations were determined in the spoils. The Ni zone also appeared to be dissipated.

The 15 m of overburden which resulted in spoils where transect T3 was located (Figure 55), contained a rather consistent 2 to 3 m thick clay zone in the strata immediately above the coal, and a 5 m thick zone of saline material near the surface. The clayey material was effectively eliminated during the spoiling process; the saline zone was not. Surficial overburden material, such as the saline zone, may not be mixed by the dragline to as strong a degree as subsurface zones. This may be due to a tendency to spoil surface materials onto the pit base in a dumping fashion, whereas, deeper overburden materials are often cast from the bucket over a larger area and slide down an existing spoil pile resulting in greater mixing.

Spoil transects T4 and T5 consisted of material that had been rehandled, i.e. moved by the dragline twice during the spoiling process, thus, the order and position of the spoils in the pit were not normal due to the rehandling. Therefore, to insure spoil samples would correspond to the inimical zones of the overburden the transects were positioned across the mine passes and parallel to each other (Figure 55).

Inimical Ni and EC zones were very small or nonexistent in overburden which resulted in spoils delineated by transects T4 and T5, however a great amount of clay material was present. The clay affected material ranged from < 10% to as much as 50-75% by volume.

The spoiling process resulted in nearly complete dilution of the small zones of Ni and EC, but a substantial volume of clay material remained in the spoil pile. The clay from the overburden remained essentially intact due to the lack of a medium with which to mix.

SUMMARY

The ability of a dragline to dilute problem overburden materials during the spoiling process was evaluated. Cross sections were prepared which depicted the distribution of several chemical and physical parameters in the overburden and the resultant spoil material as a function of dragline spoiling methods (Figures 58-62).

The proportion of problem overburden material to acceptable material was compared to the same defined proportion in spoils (Figure 63). One might expect that spoiling would decrease this ratio due to the mixing

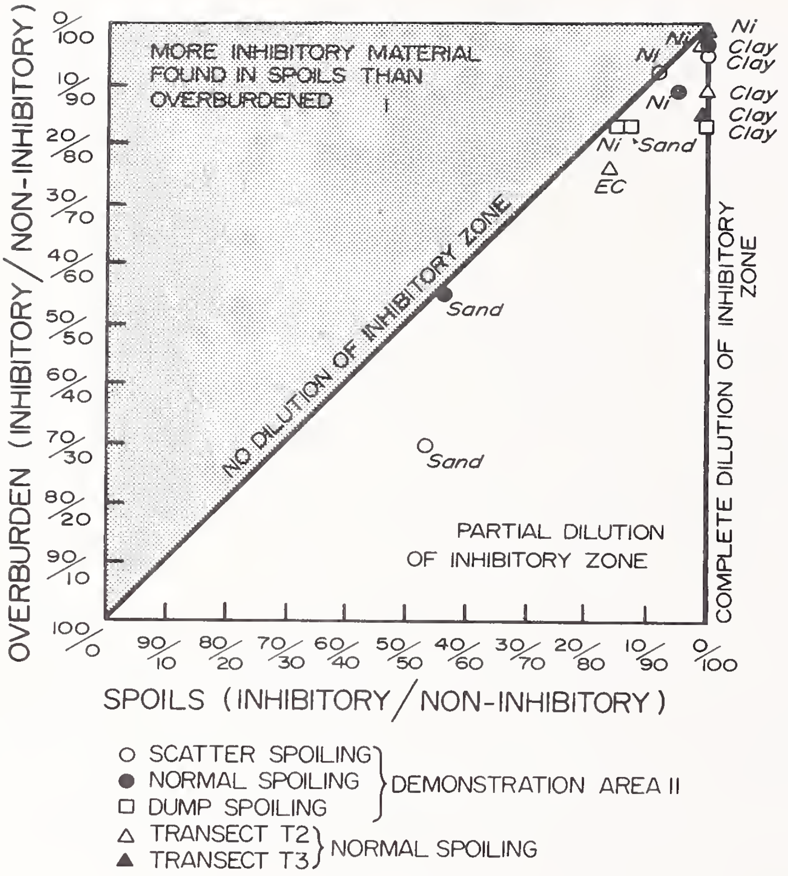


Figure 63. Relationship between volume of inhibitory material in overburden and resultant spoils.

effect. As shown in Figure 63, when the problem material constituted less than 5% of the total overburden volume, such material was essentially not detectable in the spoil pile. When the problem material constituted 5 to 15% of the overburden volume the material often could not be detected in the spoils 50% of the time; although the volume of the problem material detected in the spoils was generally less than that found in overburden, indicating partial dilution occurred. When problem material exceeded 15% of the overburden volume it was always found in the spoil pile, generally to a smaller extent, again indicating partial dilution.

There were instances when certain overburden problem zones did not appear to undergo dilution during spoiling, or in several instances (e.g. EC) these data suggested the spoils contained greater volumes of problem material than the overburden source. It is felt that such results are due in part to sampling error, also the introduction of porcelanite fill into this particular study generally produced spoils at a higher salt level than the overburden source. Such results point to the difficulties encountered when quantitative measurements are meshed with the constantly unpredictable problem bound process of dragline spoiling.

It is apparent that we cannot rely entirely on dragline spoiling methods to dilute chemical or physical overburden problems. Alternatives dealing with reclamation of inhibitory overburden materials need to be developed and selective placement during the spoiling process is one possible technique to be considered.

IX

CHEMICAL CONCENTRATION CHANGES WITH TIME IN MINE SPOIL MATERIAL

Premine studies serve to characterize the chemical framework of overburden materials so that suitable materials for reclamation can be placed in the root and aquifer zones during the mining operation. It is generally assumed that the chemical framework determined for overburden will be representative of that in the future in-place-spoil materials. However, concern exists that the interactions of overburden blasting, materials handling, material exposure to a different gaseous atmosphere (reduced to oxidized), and time could cause changes in the chemical concentration of certain elements once the spoils are in place. If true, then the chemical nature of the spoil system may be notably different than that determined during premine overburden studies.

Although a quantitative evaluation of this potential problem might best be determined through careful laboratory studies, an opportunity to help confirm or disprove this problem presented itself in the field. Near Colstrip, MT at the Western Energy Company Rosebud Mine, a 180 m long portion of overburden was carefully characterized for several chemical properties prior to being backfilled by a dragline. The backfill area was then characterized for the same chemical parameters in order to document the spoil mixing phenomena (Dollhopf et al., 1978c). In order to determine whether the chemistry of the backfilled material would undergo changes with time the material was redrilled for analysis during the following 15 months. The objective of this study was to determine whether chemical transformations occur in the backfill area which could cause notable changes in the spoil chemistry.

METHODS

The research site consisted of a 180 m long by 60 m wide backfill area (Figure 64). The backfill zone was created during January 1978 by a Marion 8050 dragline. A road was constructed on the material to permit access of the drill rig. A transect of 11 drill sites spaced 15.2 m apart was sampled with depth in 0.9 m increments in February 1978, October 1978, and April 1979. At each site the exact borehole location was offset approximately 1 m from the previous drilling date disturbance. Borehole sampling depth ranged from 11.9 to 18.3 m across the transect of 11 drill sites. Air-rotary drilling was used without the aid of solutions. Samples were analyzed for pH, EC, Pb, Ni, Zn, and particle size distribution (Appendix C).

RESULTS

The experimental area premine overburden chemistry and resultant spoil material were described earlier in Chapter VIII regarding spoil mixing phenomena. To summarize, during February 1978 the dragline spoiling operation

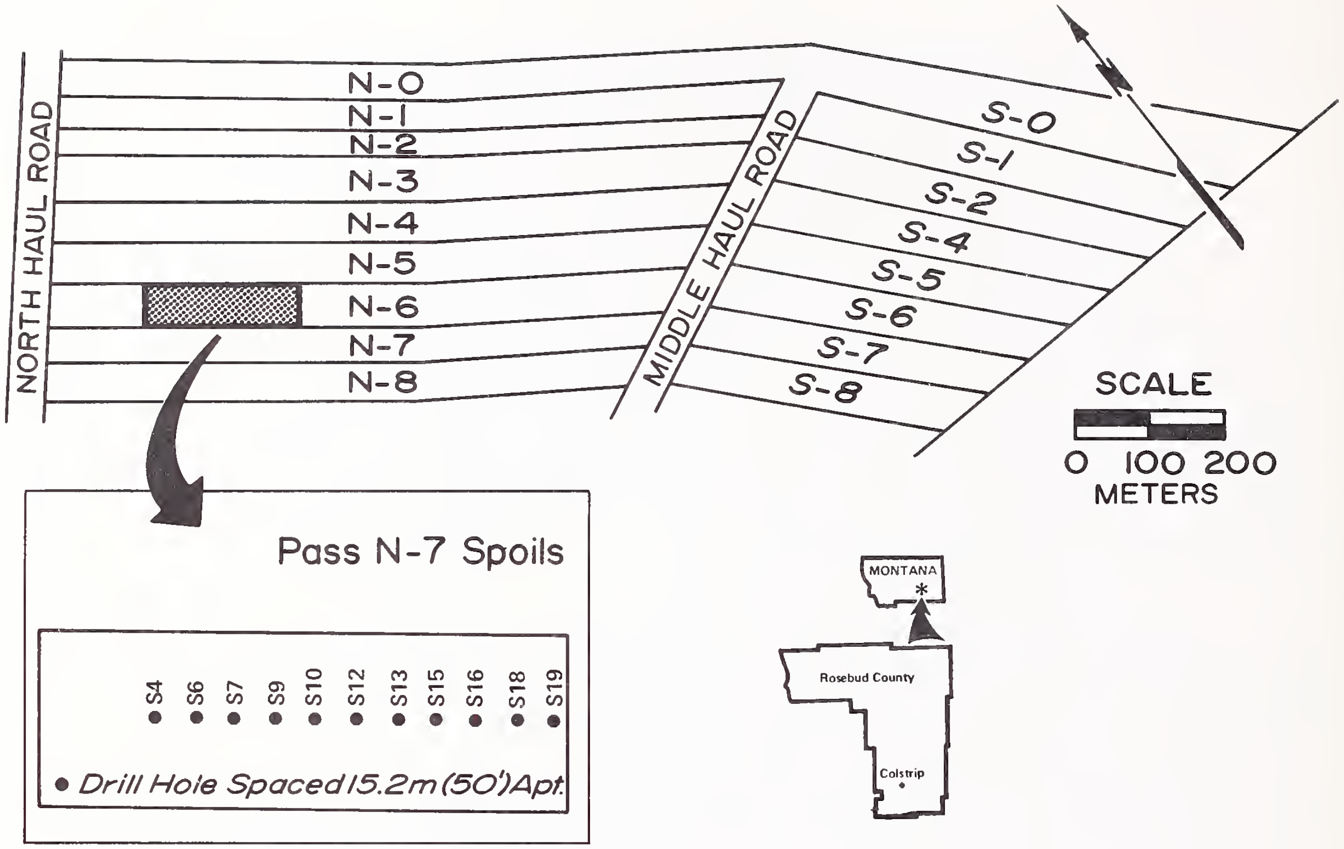


Figure 64. Location of backfill area used to evaluate spoil chemical changes over time at the Western Energy Rosebud Mine (Area B) near Colstrip, MT, during 1978 and 1979.

mixed/diluted materials sufficiently well that overburden units unsuitable for reclamation were often found acceptable in the spoil pile. When unsuitable material constituted less than 5% of the total overburden volume, such material was essentially not detectable in the spoil pile. When the unsuitable material constituted 5 to 15% of the overburden volume the material often could not be detected in the spoils 50% of the time; although the volume of the unsuitable material detected in the spoils was generally less than that found in overburden, indicating partial dilution occurred. When unsuitable material exceeded 15% of the overburden volume it was always found in the spoil pile, generally to a smaller extent, again indicating partial dilution.

It was apparent that chemical differences between newly placed spoil material and that in the premine overburden were due to physical mixing/dilution of materials, rather than to chemical transformations induced by, for example, changes in the oxidation-reduction state of spoil chemistry due to a new atmosphere.

During the 15 month period from February 1978 to April 1979 the chemistry of the in place spoil material did not change notably (Table 22). Only mean concentrations across all depths for each drill site are presented in Table 22, while complete borehole analysis results are presented in Appendix D. Regarding pH, only sites 4 and 19 from the transect of 11 boreholes had

Table 22. Mean chemical concentration found in boreholes drilled in the same backfill area on three different dates at the Western Energy Company Mine Area B near Colstrip, MT. Complete borehole chemical results are presented in Appendix D.

Borehole Site	February 1978	October 1978	April 1979
		pH	
4	7.3 ab*	7.2 a	7.4 b
6	7.3 a	7.4 a	7.4 a
7	7.2 a	7.4 a	7.4 a
9	7.3 a	7.2 a	7.5 a
10	7.4 a	7.3 a	7.5 a
12	7.3 a	7.5 a	6.9 a
13	7.4 a	7.4 a	7.4 a
15	7.4 a	7.2 a	7.4 a
16	7.1 a	7.1 a	7.2 a
18	7.2 a	7.3 a	7.2 a
19	7.4 b	7.6 b	7.1 a
		EC (mmhos/cm)	
4	3.4 a	4.5 b	2.8 a
6	3.4 a	3.2 a	4.5 a
7	3.9 a	3.3 a	2.8 a
9	5.3 b	2.9 a	2.8 a
10	4.7 b	4.7 b	3.0 a
12	4.1 b	2.9 a	2.9 a
13	3.1 a	3.0 a	2.8 a
15	3.4 b	3.0 ab	2.1 a
16	3.9 b	2.8 ab	2.1 a
18	3.7 b	2.1 a	2.0 a
19	3.6 b	2.1 a	1.6 a
		Pb (ppm)	
4	8.7 b	2.8 ab	1.6 a
6	2.6 a	1.9 a	1.1 a
7	8.7 b	1.5 a	1.0 a
9	1.8 a	1.9 a	1.5 a
10	1.6 a	1.6 a	1.2 a
12	1.8 b	1.4 ab	1.0 a
13	1.3 a	1.1 a	1.4 a
15	1.6 a	1.3 a	1.5 a
16	1.3 a	1.4 a	1.6 a
18	.8 a	1.6 b	1.0 a
19	.9 a	1.3 b	1.4 b
		Ni (ppm)	
4	.4 a	.9 b	.6 ab
6	.3 a	.5 a	.4 a
7	.6 a	.5 a	.3 a
9	.4 a	.6 a	.5 a
10	.5 a	.5 a	.3 a
12	.6 a	.6 a	.3 a
13	.4 a	.4 a	.6 a
15	.4 a	.3 a	.6 a
16	.3 a	.4 a	.5 a
18	.1 a	.5 b	.1 a
19	.1 a	.4 b	.3 ab
		Zn (ppm)	
4	1.4 a	1.6 a	1.6 a
6	.9 a	1.2 a	1.6 a
7	1.8 a	.8 a	1.7 a
9	1.1 a	1.1 a	2.0 a
10	1.1 a	1.1 a	1.9 a
12	1.5 a	1.0 a	0.9 a
13	0.8 a	0.8 a	1.7 b
15	1.9 a	0.8 a	8.4 b
16	1.2 ab	1.0 a	2.5 b
18	.5 a	1.2 b	1.1 b
19	.4 a	.7 a	3.1 a

*Means followed by the same letter within the same row are not significantly different at the 1% level using Duncan's Multiple Range Test.

significant differences between sampling dates, but such differences were still small with mean values differing by only .2 to .5 pH units.

Electrical conductance of the spoil materials generally decreased by 1 to 2 mmhos/cm during the 15 month period after the spoils were deposited. This suggests salts in the spoil zone sampled (0-18 m) were leached to a deeper depth and/or transported horizontally by the reestablished aquifer. The pit base was 20 to 28 m below the surface across the transect of drill sites and the reestablished aquifer, which was thick as 4 to 6 m above the pit base, may have leached some salts from the spoil material sampled. Precipitation during the 15 month monitoring period was 56 cm (22 in), some of which may have leached through the spoil material. It must be recognized that experimental error may have resulted in the lower salt levels in the spoils after the 15 month period. The 1 m borehole offset at each drill site for each new date may have resulted in sampling materials less saline during October 1978 and April 1979 compared to samples collected in February 1978.

Overburden materials in Montana are considered unsuitable for reclamation when Pb concentrations exceed 10 to 15 ppm (MDSL, 1977). Mean Pb concentration ranged only from 0.8 to 2.8 ppm across all sites and sampling dates, suggesting spoil pile Pb chemistry was stable with time (Table 22). During February 1978 sites 4 and 7 each averaged 8.7 ppm Pb, a high level which can be attributed to a few high values, e.g. 50 ppm, caused by sample contamination from drilling grease (see Chapter X). Although statistical significance was attained between sampling dates for a few sites (12, 18 and 19) within the 11 borehole transect, the mean differences were less than 1 ppm.

Overburden materials in Montana are considered unsuitable for reclamation when Ni concentrations exceed 1.0 ppm (MDSL, 1977). Generally, spoil pile Ni levels were not significantly different between sampling dates, suggesting Ni chemistry in this material was stable during the 15 month monitoring period. Only sites 4, 18 and 19 had significant differences between dates regarding Ni levels, mean differences were small (0.5 ppm) and no trend, increasing or decreasing, was present among these data. Again, sampling error due to borehole offset between sampling dates could have introduced differences of 0.5 ppm into these data.

Overburden materials in Montana are considered unsuitable for reclamation when Zn concentrations exceed 40 ppm (MDSL, 1977). Mean values for Zn across all sites and sampling dates never exceeded 8.4 ppm, and rarely exceeded 2.0 ppm. Levels of Zn increased in the spoil pile during the 15 month monitoring period in 9 of the 11 boreholes, and significantly so in borehole sites 13, 15, 16 and 18. These increases were quite small averaging approximately 1 ppm. Realizing the margin of error associated with field sampling methods, it is possible these differences were due to error rather than an enhanced release of available Zn into the spoil system.

Table 23 presents linear correlation coefficients (r) between different sampling dates regarding borehole chemistry with depth. All regression coefficients were low, indicating the 1.0 m borehole offset for successive sampling dates resulted in collecting materials chemically dissimilar. Drilling the loose-unconsolidated spoil material with the air-rotary methods may have at times resulted in deep samples being intermittently mixed with zones nearer the surface as the hole wall either eroded or caved-in. This mechanism could, in part, cause poor chemical correlations with depth between boreholes separated by only 1.0 m. Although these correlation coefficients suggested sampling error was notable between dates at each site, it must be realized that chemical concentration differences between depths and dates at each site was small, and as previously discussed, the spoil chemistry was largely stable during the 15 month monitoring period. Low regression coefficients were attained because on a depth by depth comparison, values could not be reproduced in a consistent manner across sampling dates.

SUMMARY

Chemical characterization was performed on a 180 m long section of overburden, the resultant spoils, and the same spoils after 15 months of time had elapsed. The objective of the study was to determine whether the spoil chemistry differs notably from premine assessment of the overburden chemistry.

Chemical differences between newly backfilled spoils and that in the premine overburden were due largely to physical mixing/dilution of materials rather than to chemical transformations. Once the spoils were in place, the chemical status of materials did not change notably for 15 months regarding the parameters pH, EC, Pb, Ni, and Zn. Clearly, toxic zones were not being generated in the spoil material. These data did indicate spoil salinity generally decreased by 1 to 2 mmhos/cm and Zn levels increased by 1.0 ppm during the 15 month period, however such trends were not conclusive.

It has been determined that in the Northern Great Plains the reestablished aquifer in a mine pit generally contains water with higher salinity than did the premine aquifer (Van Voast et al. 1978). Data in this chapter suggest the increased salinity measured in postmine aquifers may be due in part to leaching of salts from spoil materials.

Table 23. Correlation coefficients between different sampling dates regarding borehole chemistry in spoil material at the Western Energy Company Mine Area B near Colstrip, MT.

Borehole Site	Linear Correlation Coefficients (r)		
	Feb. 1978 vs Oct. 1978	Feb. 1978 vs April 1979	Oct. 1978 vs April 1979
		pH	
4	.27	.36	.80
6	-.12	-.38	-.13
7	-.23	-.49	.30
9	-.20	-.20	.46
10	x	x	x
12	-.06	.06	.07
13	-.62	.10	.17
15	.49	-.12	-.38
16	.06	-.13	.09
18	.25	.25	.21
19	-.18	-.48	-.15
		EC (mmhos/cm)	
4	.45	-.08	.36
6	-.15	-.08	-.01
7	.16	.08	.17
9	.05	-.05	-.37
10	x	x	x
12	.10	-.20	-.28
13	-.26	-.55	.26
15	.13	-.09	.31
16	.44	.40	-.43
18	.02	.27	-.15
19	-.15	-.16	.58
		Pb (ppm)	
4	.04	.12	-.43
6	.05	.41	.21
7	.04	-.16	-.47
9	.48	-.01	-.18
10	x	x	x
12	.12	.44	.10
13	-.44	-.17	.29
15	.13	.00	-.10
16	.05	.59	-.26
18	.37	.12	-.11
19	-.31	.24	-.01
		Ni (ppm)	
4	.57	-.38	-.53
6	-.17	-.16	.24
7	.64	-.51	.45
9	.36	.27	.07
10	x	x	x
12	-.08	.41	.12
13	-.14	.12	.19
15	.03	-.20	-.39
16	.13	.45	-.33
18	-.14	-.02	.07
19	-.18	.20	-.46
		Zn (ppm)	
4	.54	-.22	-.16
6	-.16	-.08	.21
7	.08	-.29	-.39
9	-.24	-.22	-.01
10	x	x	x
12	.07	.50	-.08
13	-.20	.19	.46
15	.22	.24	-.26
16	-.20	.22	.01
18	.29	.47	-.15
19	-.23	-.20	-.11

[x] Not Determined

X

OVERBURDEN SAMPLE CONTAMINATION FROM DRILL STEM JOINT LUBRICANT

BACKGROUND

In order to secure necessary permits to operate a coal strip mine in the western coal region, overburden samples must be collected and analyzed for chemical properties. These premine overburden analyses indicate which stratigraphic intervals are most suitable for reclamation. Included in these analyses are DTPA extractable Pb, Zn and Cu, which are generally present in the low part per million concentration range. At such low concentrations, the potential for sample contamination is large from either field or laboratory error. The diethylenetriamine pentaacetic acid (DTPA) extractable trace element concentration is representative of that concentration which is available to plants.

Powers and Sandoval (1976) found that drilling technique can introduce significant error in overburden chemical analyses. They found that rotary drilling with air and no transport solutions, which produced a sample composed of chips, resulted in the most accurate chemical analyses. All methods of coring, which used various solutions, resulted in overburden sample analysis with greater error. Their evaluation was based on analyses for electrical conductivity, calcium, magnesium and sodium. Trace elements were not included, however.

This research project on selective placement of overburden materials included construction of nearly 400 boreholes, all of which were drilled with a rotary system without aid of solutions. Preparation of cross-sectional diagrams which depicted the chemistry in both overburden and spoils indicated the high likelihood of trace element contamination within samples due to drill stem joint lubricant. For example, Figure 62 shows the cross-sectional distribution of DTPA extractable Pb in overburden and resultant spoils (from Dollhopf et al., 1978b). Notice the high concentration (15 to 170 ppm) of Pb in spoils near the 985 m elevation, approximately 19 m below the surface. This spoil zone contained more unsuitable lead than did the original overburden, suggesting sample contamination. Also the elevation of this toxic zone varied in an identical manner to the surface, suggesting a drilling technique induced phenomena. Other zones of Pb toxicity in the spoil cross section are generally found in increments of 6 m from the surface, which correlated to the drill stem length. The drill stem joint lubricant used for boreholes shown in Figure 62 had a lead additive. Other cross sections showing distribution of DTPA extractable Zn indicated similar incremental zones with depth for Zn toxicity when the grease used had a Zn additive.

The suspected contamination source was the grease application within threaded joints of 6 m long sections of drilling pipe (Figure 65). To facilitate disassembly of drilling stems and drilling bits when a borehole has been completed, a joint lubricant was applied to the threaded stem ends as the sections were assembled during the drilling. A minute portion of the grease was apparently released into the drilling pipe and was forced out the drill bit by air from the compressor, resulting in overburden chip samples being mixed with the grease.

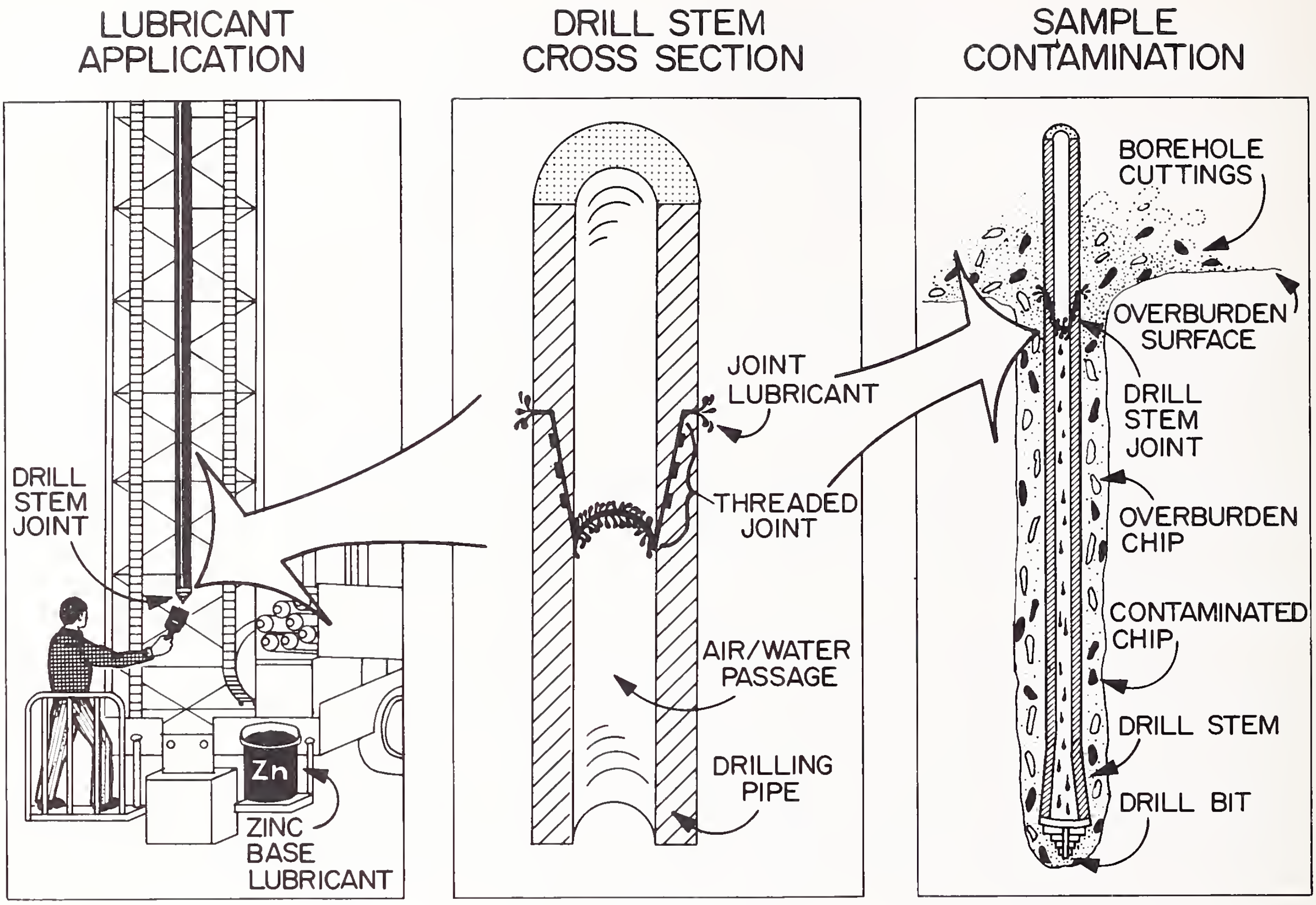


Figure 65. Illustration of overburden sample contamination from drill stem lubricants.

Drill stem joint greases perform two main functions; 1) the petroleum base acts as a sealent on each joint so air and drilling solutions moving through each pipe stem do not leak out, and 2) the additives such as metal flakes resist galling or metal wear by serving as a physical barrier between metal surfaces and these metal flakes also resist excessive torque or tightening of the joint. Additives to the base material produce a grease with unique characteristics, e.g. high temperature stability, suitable for special applications. The petroleum base grease generally used for overburden drilling applications contains Zn, Cu or Pb metal flake additives since they optimize the grease performance.

This contamination problem is apparently widespread and is only beginning to be recognized. In discussions with regulatory agencies[1] and coal companies[2] instances were cited where certain overburden trace element analyses were questionable, and such erroneous data generally were attributable to the type of joint grease used during drilling. To assess the impact of the various joint lubricants in use, a laboratory experiment was designed to determine the

---

[1] Personal communication with Mr. N. Harrington, State Lands Dept., Reclamation Division, Helena

[2] Personal communication with Mr. G.M.L. Robinson, Senior Hydrologist, Western Division, Peabody Coal Company, Denver, Colo.

DTPA extractable heavy metal concentration of the greases, and to determine the amount of a specific joint compound necessary to contaminate an overburden sample.

METHODS

Samples of several drill stem joint lubricants typically used during overburden sampling programs were obtained from Bestolife Corporation[√] (Table 24).

Table 24. Total concentration of dominant trace elements in greases used to lubricate threaded portions of drilling pipe.

Grease Description		Total Concentration (% by weight)[*+]				
Base	Additives	Pb	Zn	Cu	Mo	C_2F_4
Petroleum	Pb	60%				
Petroleum	Zn		50%			
Petroleum	Pb, Cu	10%		10%		
Petroleum	Zn, Mo, C_2F_4(teflon)[†]		30%		1%	15%

[*] Recall this is total elemental concentration which is a greater amount than that extracted with H_2O or DTPA.

[+] All data supplied by manufacturer (Bestolife Corp., Box 6195, Dallas, Texas).

[†] Tetrafluoroethylene (C_2F_4) commonly know as teflon.

A series of grease-sand mixtures were formulated and analyzed using DTPA extraction procedures and atomic absorption techniques (Soltanpour et al., 1976; Lindsay and Norvell, 1969). A sand soil (98% of all particle diameters between .05 and 1 mm) was used as the diluting medium for the grease. The sand was dried to an air dry state and passed through a 1 mm sieve prior to analysis. A total of six subsamples of the sand were analyzed to determine baseline DTPA extractable concentrations for elements of interest (Pb, Ni, Zn, Mn, Cu and Cd).

Each of the four greases being investigated was treated in the following manner. A 1.0 g subsample of the grease was measured into a glass beaker to which 50.0 g of sand were added. The sand and grease were mixed by hand using a stainless steel spatula until a uniform consistency was obtained. From this mixture, three 10.0 g subsamples were measured into glass beakers for analysis by DTPA extraction procedures. A 1:51 dilution of grease to soil was obtained in this manner for each of the four greases. This procedure was duplicated for each grease.

The petroleum base grease with Zn additive (50%) was selected for a more intense analysis. A series dilution of grease to soil was implemented. First a 1:50 grease to sand mixture was prepared and used as the source to prepare five other dilutions; 1:5000, 1:2500, 1:1250, 1:500 and 1:250. These ratios

---

[√] Reference to specific brands, equipment, or trade names in this report is made to facilitate understanding and does not imply endorsement by either the Bureau of Mines or the Montana Agricultural Experiment Station.

were produced by mixing 0.5, 1.0, 2.0, 5.0 and 10.0 g of the 1:50 mixture with sufficient sand to total 50.0 g, respectively. From each of the five dilution mixtures three 10.0 g subsamples were analyzed for DTPA extractable element concentrations. The dilutions 1:2500, 1:1250, 1:500, and 1:250 were prepared and analyzed six times from newly prepared 1:50 mixtures, yielding 18 replications. In addition a 1:100 dilution mixture was prepared by mixing 1.0 g of grease with 99.0 g of sand, from which three subsamples were analyzed for DTPA extractable element concentrations.

PETROLEUM BASE GREASE EVALUATION

DTPA is an amino acid that has a strong affinity for metal ions, and will readily arrange its molecular structure in a clawlike chain around metal ions. This resulting *chelate* is very stable, and the metal ions cannot be precipitated. Therefore DTPA is a useful chemical for extracting trace metals from soils.

DTPA, although an effective chelating agent, has finite extraction capabilities. Therefore, if the extractable metal (M^+) or several metals (M_1^+, M_2^+, M_3^+, M_4^+) in a sample are present in excess of the chelating ability of the DTPA extraction solution, the extractable M^+ concentrations from the sample will remain approximately constant regardless of the amount in excess. During chelation, DTPA demonstrates some tendencies toward preferential extraction. For example, DTPA mole fraction diagrams presented by Sommers and Lindsay (1979) predict the preferred chelation of Pb when Pb and Cu are present in the sample. These DTPA behavior characteristics are important in the following discussion.

DTPA was capable of extracting the dominant metals of each drill stem grease investigated, plus various amounts of secondary metals. For example, the grease with a Pb additive was composed of 60% Pb. When mixed with soil (1:51), the DTPA extraction indicated 2083 ppm Pb, and small amounts of Zn, Cd, Mn and Cu were present (Table 25). Similarly the grease with Zn additive

Table 25. Mean and standard deviation of DTPA extractable trace metal concentrations for a sandy soil and for grease:sandy soil (1:51) mixtures.

Sample Description	DTPA Extractable Concentrations (ppm)					
	Pb	Ni	Zn	Cd	Mn	Cu
Sandy Soil[*] n = 3	.3 ± .1	.2 ± .1	.3 ± .1	.03 ± 0	2.3 ± .2	.09 ± .01
Petroleum Grease Additive	Mixtures of Sand Soil with Grease					
Cu, Pb (n[+] = 6)	687 ± 48	.2 ± .1	31 ± 3	.03 ± .01	6.1 ± .1	57 ± .01
Pb (n = 6)	2083 ± 30	0 ± 0	3.5 ± 2	.04 ± .01	.5 ± .1	.03 ± .01
Zn (n = 3)	1.9 ± .4	.1 ± .1	760 ± 26	.36 ± 0	.5 ± .2	0 ± 0
Zn, Mo, C_2F_4 (n = 6)	.4 ± .1	.1 ± .1	5.1 ± 2.4	.01 ± 0	1.4 ± .1	.15 ± .01

[*] 98% of all particle diameters between .05 and 1 mm

[+] n indicates the number of values per mean was composed of 50% Zn. When mixed with soil (1:51) the DTPA extraction resulted in 760 ppm Zn, and small amounts of Pb, Ni, Cd and Mn were present. Another petroleum base grease was composed of 15% teflon (C_2F_4), 30% Zn and 1% Mo. When mixed with soil (1:51) the DTPA extraction resulted in only 5:1 ppm Zn, and much smaller quantities of Pb, Ni, Cd, Mn and Cu were present. Molybdenum and flouride concentrations were not determined, however. The grease with Cu and Pb additives was composed of 10% Cu and 10% Pb. When mixed with soil (1:51) the DTPA extraction resulted in 687 ppm Pb compared to only 57 ppm Cu, a tenfold difference. In addition smaller quantities of Zn and Mn were present along with trace amounts of Ni and Cd. The apparent preferred extraction of Pb over Cu by DTPA substantiates the findings of Somers and Lindsay (1979).

These data clearly indicate that DTPA will extract metals from both soil material and grease contaminates. When petroleum base greases with metal additives were mixed (1:51) with soil the DTPA extractable concentrations for Pb (687 to 2083 ppm) and Zn (760 ppm) far exceeded the toxic criteria for Pb (>15 ppm) and Zn (>40 ppm) in overburden material. Overburden sample contamination from grease with Pb additives has been observed to range from 15 to 170 ppm Pb (Figure 66). Similarly, sample contamination from grease with Zn additives has been observed to range from 40 to 100 ppm. It is apparent that far less than 1 g of grease in 50 g of soil will contaminate the sample to the point that the sample could be falsely interpreted as toxic in nature. Although the data in Table 25 do not contain all trace elements of concern in mine land areas, it seems probable that only Pb could produce significant contamination in soil samples when using the petroleum base grease with Cu and Pb additives, and only Zn could produce significant contamination when using the base grease with Zn additive. Other trace elements in these greases would not introduce significant error. The grease with teflon, Zn and Mo additives did not create significant error regarding other trace elements analyzed (Table 25), although Mo and F were not analyzed. Therefore this grease may be the most desirable from a chemical standpoint, compared to the others tested.

GREASE:SOIL SERIES DILUTION

When the petroleum base-Zn additive grease was incrementally diluted with soil, the results indicated that at low dilution ratios (1:50, 1:100, 1:250) the amount of Zn present was so great that the DTPA chelation sites were saturated. This resulted in nearly identical DTPA extractable concentrations for Zn (776 ppm) at all three levels of dilution (Figure 12). As the quantity of grease in the soil decreased (<1:250) the amount of DTPA extractable Zn decreased. Even when the grease to soil ratio was 1:5000 the DTPA extractable Zn concentration was 146 ppm, well in excess of the overburden inhibitory level (40 ppm) established in Montana by the Department of State Lands (1977).

During our overburden investigations, when sample contamination was attributable to the presence of a drilling grease with Zn additive, the DTPA extractable concentration was on the order of 40 to 100 ppm Zn. As shown in Figure 66, there was only 1 part grease to 10,000 or even 20,000 parts soil to attain DTPA extractable concentrations on the order of 40 to 100 ppm Zn. Thus a very minute amount of grease can cause significant sample contamination resulting in false interpretation of overburden characteristics.

The trace element concentration of Pb, Ni, Cd, Mn and Cu were also determined in the series dilution using the grease with Zn additives. The concentration of these trace elements behaved in a similar and unique manner as a function of the dilution series. For example, while the Zn base grease saturated the DTPA chelation sites at low grease to soil dilution ratios (1:50, 1:100, 1:250) the amount of DTPA extractable Pb remained nearly constant at about 2 ppm (Figure 66). This suggests that if a large amount of petroleum base grease with Zn additive were present in an overburden sample (>1:250), the dominance of Zn on chelation sites would mask out the ability of DTPA to chelate other trace elements. Thus, the sample analyses may not be valid for any trace element extracted by DTPA. When less Zn base grease was mixed with soil (<1:250), the chelation sites were not as strongly dominated by Zn and the amount of DTPA extractable Pb increased to 4 ppm (Figure 66). As even less Zn base grease was mixed with soil the amount of DTPA extractable lead steadily decreased and approached the baseline extractable Pb concentration in the soil material (.3 ppm). In addition to the suppression of extractable Pb when Zn dominated the chelation sites, other trace elements present in the mixture such as Ni, Cd, Mn and Cu had their extractable concentrations suppressed in a similar manner to Pb.

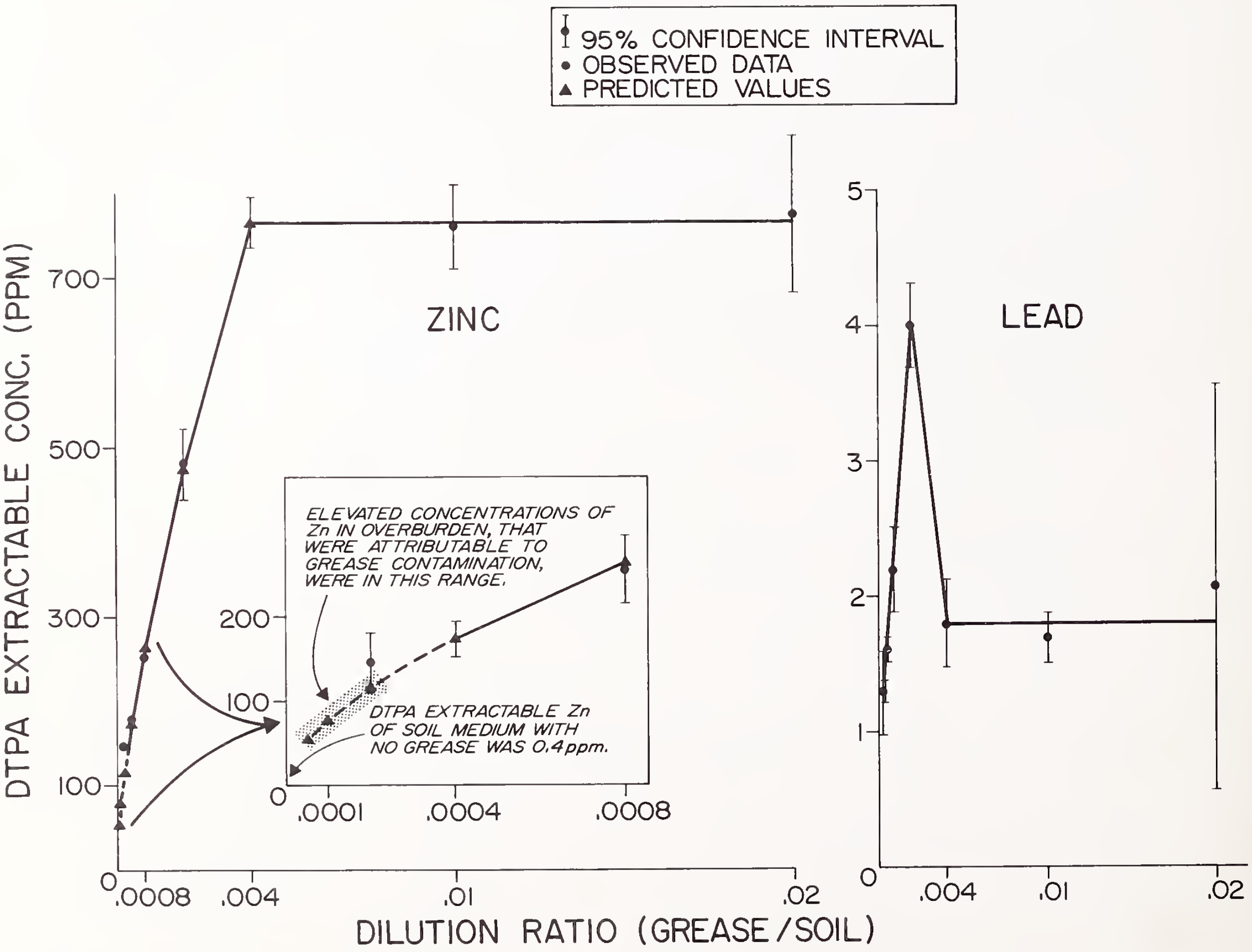


Figure 66. DTPA extractable Zn and Pb from mixtures of a petroleum base grease (50% pure Zn) with soil.

SUMMARY

Drill stem lubricants can contaminate DTPA extractable trace element analyses. DTPA will extract metals from both soil material and grease particles introduced into the soil during the drilling operation. The presence of drilling greases in overburden samples can produce errors as much as several hundred parts per million in trace element analyses. When the metal within a grease base, e.g. Zn, contaminates the Zn content of an overburden sample the error may be apparent since the concentration may be 100 times adjacent samples. However, the dominance of Zn on the DTPA extracting solution chelation sites may suppress the extraction of other trace elements which could produce error in overburden characterization studies. Sometimes a grease with a metal additive, e.g. Zn, has been assigned for drilling overburden with the understanding that subsequent sample analyses for Zn are not needed since the overburden has been characteristically low in Zn content. It has been assumed that other DTPA extracted trace elements will be unaffected by the grease. Because of the suppression effect of Zn upon the DTPA chelation process all trace element analyses of DTPA extraction become questionable.

The three petroleum base greases in general use by drillers which contain metal additives (Pb, Zn, Cu) all caused significant trace metal contamination of overburden samples. The fourth petroleum base grease studied had teflon and metal (Zn, Mo) additives and did not cause significant sample contamination. Molybdenum and flouride were not analyzed, however. This suggests that the presence of teflon in the grease may have made the metal additives less available to extraction by DTPA.

The mineral talc ($Mg_3Si_4O_{10}$ $(OH)_2$) in a powder form is a suitable substitute for Pb, Zn, Cu and Mo metal additives[1/]. Talc as an additive provides the petroleum base with anti-galling and anti-torque characteristics similar to Pb, Cu and Zn metal additives. Generally, less talc would be required in the grease (15% by weight) compared to metal additives (10-50% by weight)[1/], thus the chance of sample contamination is less. Talc is composed of 19% magnesium by weight. Since magnesium levels in soils are quite high the potential for sample contamination in this element is low. The presence of teflon in addition to the talc may provide the optimum combination for a drilling lubricant that will not contaminate soil samples. The lubricating performance of the petroleum base grease with talc and teflon additives is less desirable than with metal additives, but should be suitable for the relatively shallow boreholes constructed in mine land overburden studies. At the present state of knowledge, it is recommended that a petroleum base grease (e.g. type ANLGI-1)[1/] with talc and teflon additives (each 15% by weight) be utilized when sampling overburden for chemical analysis, rather than metal additives such as Zn, Pb, Cu and Mo. Such a mixture is not commercially available but the components can be obtained for custom mixing. If it is planned to analyze flouride in overburden samples, then teflon (C_2F_4) should be deleted from the grease because of potential contamination. Generally flouride is not analyzed. Further testing of the petroleum base grease with talc and teflon additives is also recommended.

[1/] Personal communication with Mr. D. Prengleman, RSR Corporation, Box 6195, Dallas, Texas

XI

PREDICTING POST-MINE GROUND-WATER TRACE METAL CONCENTRATION

One major factor that controls the chemical content of ground water is the chemistry of its flow medium (Moran, et al., 1978). It would be desirable to be able to predict ground-water quality in mine spoils from overburden stratigraphy characteristics. Overburden intervals which result in desirable ground-water quality could be backfilled into the aquifer zone, while undesirable intervals could be selectively isolated. Identification of desirable or undesirable overburden materials for aquifer reestablishment remains a technical problem.

At present, western states do not require tests on premine overburden samples to predict water quality that would result in spoil aquifers. Investigators have suggested techniques to predict post-mine ground-water quality. Van Voast et al. (1978) leached columns filled with spoil material from southeastern Montana and determined that leachates were similar to spoil waters sampled in the field. They also determmined that a water extract from a saturated soil paste (U.S.D.A., 1969), simpler method than column leaching, was comparable to the leachate from a column. Their evaluation was based on the parameters Ca, Mg, Na and specific conductance. Most western states require determination of Ca, Mg, Na and specific conductance on saturated extracts from overburden samples in order to evaluate the salt and sodium hazard regarding plant establishment. Thus, these same data may be suitable to evaluate soluble salt status of ground water in the post-mine aquifer.

Procedures to predict trace element levels in post-mine aquifer remains a topic that is not clearly defined. In western states, overburden analyses for Pb, Ni, Zn, Mn, Cd and Cu are in general accomplished using the DTPA extraction method (Soltanpour et al., 1976; Lindsay and Norvell, 1969). This technique is used to determine plant available metal concentrations in soils since research has shown good correlation between DTPA extractable metal levels and plant response (Lindsay and Norvell, 1978). DTPA is a much stronger extracting solution than water and would result in values ten times or more greater than those trace element levels in ground water. Although DTPA will extract larger amounts of trace elements than water, the amount may be proportional to that extracted by water. The existence of such a statistical relationship could provide a prediction of post-mine aquifer quality based on DTPA analysis procedures already in general use.

In this study, attempts were made to determine the correlation between DTPA and water extractable trace elements in overburden samples. In addition, the feasibility of using leaching columns to predict post-mine ground-water quality for several trace elements was determined.

METHODS

Twenty-five overburden samples from the Western Energy Rosebud Mine Area B near Colstrip, Montana were air-dried, fragmented, passed through a 1.0 mm sieve and thoroughly mixed prior to analysis. From each sample, triplicate

10 g subsamples were analyzed for Pb, Ni, Zn, Cd, Mn, and Cu using the DTPA extraction method (Soltanpour et al., 1976, Lindsay and Norvell, 1969).

From these same 25 samples, a 50 g subsample was packed into glass columns for leaching (Figure 67). Each column was saturated with 20 ml of distilled water, and the saturated soil system was allowed approximately 2 hours to equilibrate. Then a second 20 ml portion was added, resulting in approximately a 20 ml leachate sample which was collected in a glass flask. The filtrate was acidified with concentrated HNO_3 to reduce adsorption of trace metals to the container wall.

Furnace atomic absorption was used for the Pb analysis in the leachate samples because of both very low Pb concentrations and inherent low sensitivity of flame atomic absorption. Standard flame atomic absorption procedures were used for the other elements.

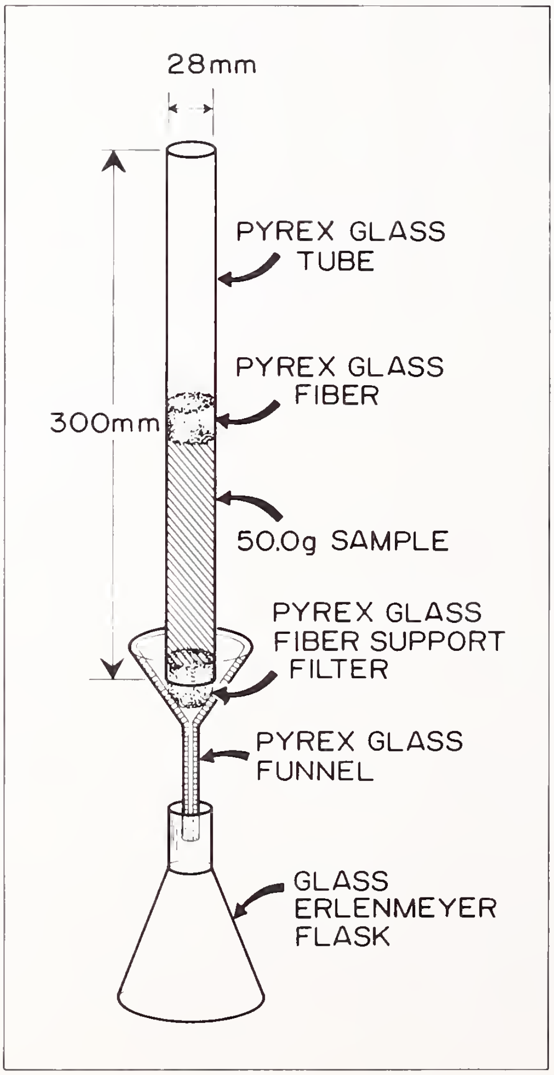
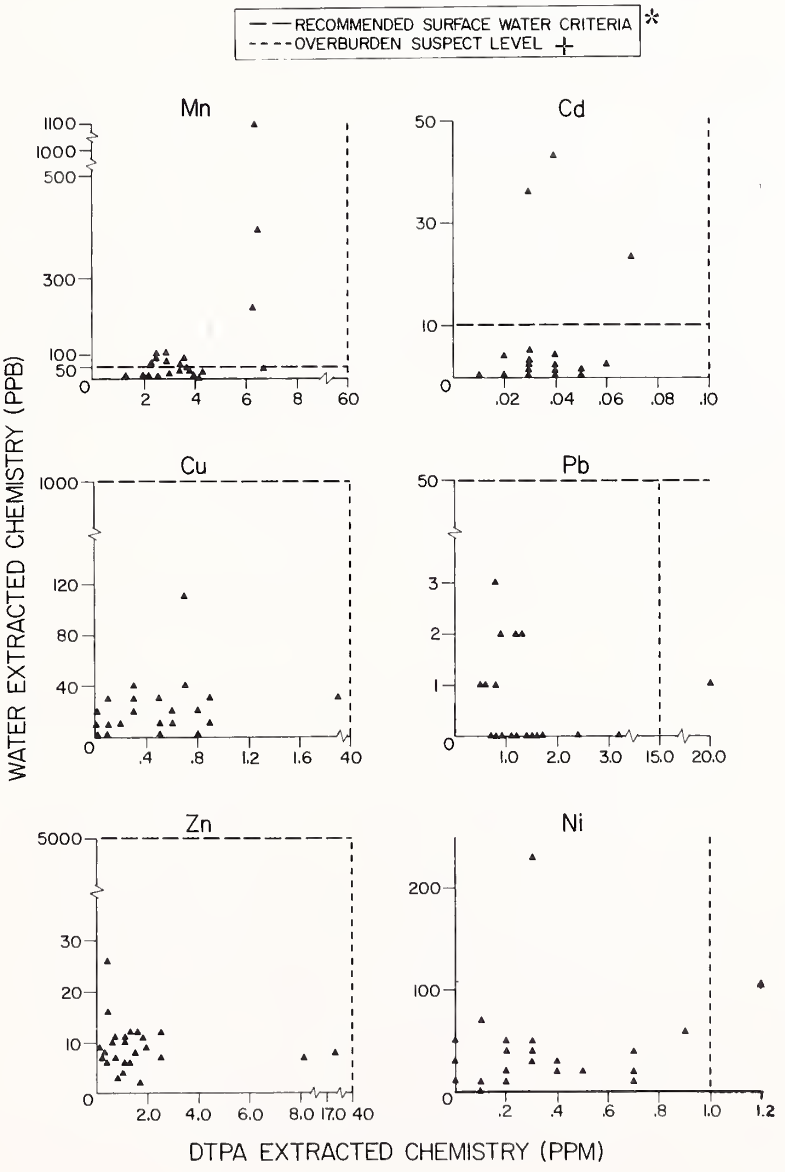


Figure 67. Water leaching apparatus.

DTPA EXTRACT:WATER EXTRACT RELATIONSHIPS

Figure 68 compares the DTPA extractable concentration to the column water leachate concentration for Mn, Cd, Cu, Pb, Zn and Ni in 25 overburden samples. Linear correlation between DTPA and water extractable trace element concentrations was very poor (Table 26). This suggests that the DTPA extractable trace metal concentration of overburden material bears no relationship to the chemical characteristics which that material may give to ground water if placed in an aquifer zone. This results from the differing chelation strengths of DTPA for various metals and the differing solubilities of metals extracted by water.



*U.S. Dept. Health, Education, Welfare (1962)

+Montana Dept. of State Lands (1977)

Figure 68. Comparison of DTPA extractable and water extractable (leaching column) trace metal concentrations in 25 overburden samples.

Table 26. Correlation coefficient between DTPA extract concentration and water leaching column extract concentration for some trace elements. An (r) of 1.0 indicates a perfect correlation.

Correlation Coefficient (r)					
Pb	Ni	Zn	Cd	Mn	Cu
0	.05	-.11	.24	.57	.24

SOIL WATER COLUMN LEACHATE RELATIONSHIP

Numerically, the DTPA extract concentrations were greater than values obtained from leaching column water extracts, up to 1000 times in many instances (Figure 68). The 25 overburden samples used in this study came from a research site where numerous observation wells were present. If the leachate chemistry from this column method has merit as being representative of the *in situ* post-mine ground-water chemical system, then the leachate chemistry should be comparable to water from observation wells. In all tests, resulting leachate qualities were found to be similar to qualities of ground waters sampled in the field. For example, the Mn concentration in leachate samples exceeded the recommended drinking water standard (.05 ppm) in nearly 50% of the samples (Figure 68). This was comparable to ground water in overburden (Table 27). Ground water in spoil material always exceeded the recommended drinking water standard for Mn, however. Likewise, about 10% of the leachate samples and field ground-water samples exceeded the recommended level for Cd in drinking water. Waters from both observation wells and column leachate samples never exceeded the recommended drinking water standards for Cu (1 ppm), Pb (.05 ppm) and Zn (5 ppm) (Table 27).

Examination of the water column extracted chemistry and corresponding water well chemistry indicate that leachates from columns may provide predictions of post-mine water quality. It should be clearly noted that the statistical means, and ranges for these comparisons between column leachates and waters from wells often differed by as much as a factor of ten (Table 27). Part of this difference can be attributed to sampling and lab error. Certainly the leaching column technique is not a perfect simulation of the *in situ* ground-water system, so the technique itself introduces error. However trace element concentrations in column leachates were comparable to concentrations in waters from wells to a degree which indicates the potential exits to judge which overburden materials would be most suitable for aquifer reestablishment. However, comparisons of water leachate metal concentrations and *in situ* ground-water quality would have to be correlated at many mines with contrasting chemical conditions to verify the usefulness of this method. Column leaching techniques are slow, expensive, and may not be practical for premine studies. An investigation should be initiated to determine whether a water saturated paste extract, which is performed on a routine basis, could provide similar trace metal levels as those attained from column leachates.

Leachate collected from a column filled with overburden may predict post-mine water quality, if that material were placed in an aquifer. Such predictive ability would allow the mine operator to evaluate alternative dragline or truck-shovel spoiling techniques to prevent placement of undesirable materials into the future aquifer zone.

Table 27. Comparison of trace element concentration (ppm) in lab column leachates and in waters from wells in overburden and spoils at the Western Energy Rosebud Mine near Colstrip, Montana.

Trace Metal	Soil Water Column Leachate Concentration (n = 25)		Observation Well Data				Recommended Maximum Permissible Concentration in Public Water Supplies[°]
			Overburden[+] (n = 16)		Spoils[Δ] (n = 10)		
	Mean ± S_x	Range	Mean ± S_x	Range	Mean ± S_x	Range	
Mn	.10 ± .22	(<.01-1.09)	.077 ± .074	(.055-.252)	.424 ± .129	(.272-.710)	.05
Cd	†	(<.001-.043)	†	(<.005-.014)	†	(<.005-.020)	.01
Cu	.021 ± .022	(<.01-.04)	.023 ± .010	(.005-.038)	.033 ± .019	(.012-.075)	1.0
Pb	†	(<.001-.003)	†	(<.01-.025)	†	(<.01-.01)	.05
Zn	.009 ± .005	(.002-.026)	.093 ± .112	(<.005-.377)	.220 ± .241	(.022-.712)	5.0
Ni	.040 ± .047	(<.01-.23)	.013 ± .006	(<.01-.018)	†	(<.01-.01)	-

[+] Wells 7-0, 15-0, 28-0, 29-0, 30-0, 32-0, 33-0 (see Figure 1 for locations)

[Δ] Wells CR-2, D-2 (see Figure 1 for locations)

[†] Many values were reported as less than (<) the indicated range, indicating a concentration less than technique sensitivity

[°] U.S. Dept. Health, Education, Welfare (1962)

XII

CONCLUSIONS AND RECOMMENDATIONS

OVERBURDEN CHARACTERIZATION

(1) Northern Great Plains coal overburden was deposited in a fluvial depositional system. These stream laid materials composed of channel, bank and flood basin deposits each had unique physiochemical characteristics. As a result, the overburden physiochemical environment is highly variable over short distances, making identification of materials unsuitable for reclamation difficult.

(2) Knowledge of the geochemical environment and depositional system may reduce overburden characterization costs by allowing concentration of the drilling and sample analysis effort to more specific areas or depths.

(3) Statistical analysis on nearly 300 boreholes in the research area indicated overburden could be characterized with a 90% accuracy only when drill sites were spaced approximately 40 m apart. Presently western states recommend an overburden sampling scheme with drill holes 600 to 1500 m apart, representing a 50% accuracy. It is probable that most unsuitable overburden materials are not being detected, resulting in such materials being unknowingly deposited in the aquifer reestablishment and root zones.

(4) Sampling overburden in a two step fashion may provide a desirable compromise between accuracy and costs. It is recommended that the overburden be sampled first with a 600 m grid intensity. This would not provide good accuracy for delineating the location and extent of unsuitable materials, but such a sampling scheme would very likely identify which overburden physiochemical parameters are present at unsuitable levels at the mine site. Then the entire area should be drilled intensively to attain a characterization accuracy of 80 to 90% (60 m grid), but these samples only need to be analyzed for those parameters which were present at inhibitory levels during the initial reconnaissance.

(5) Implementing selective handling of unsuitable overburden based on physiochemical data from drill holes 600 to 1500 m apart would be ineffective in increasing the reclamation potential of the project area since the geochemical distribution of unsuitable materials would be largely unknown.

(6) Additional research is needed regarding adequate overburden sampling intensity in western states.

SELECTIVE OVERBURDEN HANDLING

(7) Assuming accurate identification of unsuitable materials has been performed, it is possible to selectively bury such materials with a dragline as these overburden zones are encountered. This was successfully demonstrated at the research site with dragline operation. Unsuitable overburden materials would be encountered intermittently, so normal spoiling procedures would need to be interchanged with selective handling procedures during the life of the mine.

(8) Selective burial of unsuitable overburden materials could increase costs of the entire mine operation by approximately 12%. If an impermeable clay cap needs to be constructed over the burial material, costs of the entire mine operation could increase by approximately 53%.

(9) Construction of a clay cap over selectively buried overburden materials may not be needed in semiarid western states. After three years, only small quantities of water flowed from the surface to the buried material and the soil water-leaching regime associated with capped buried material was not different from uncapped buried material. If the burial site is beneath a hill top or slope a clay cap is probably not required. If the burial site has to be constructed in a valley bottom, or other land morphological feature that would collect water, then a clay cap may be advisable.

(10) When an aquifer is present in the mine pit to be excavated, knowledge of the pre- and post-mine water levels is essential for successful burial of unsuitable overburden materials. In this manner the post-mining saturated zone can be filled with desirable overburden materials, and undesirable material placed well above the aquifer, yet below the future root zone.

SPOIL MIXING

(11) When inhibitory material constituted <5% of the total overburden volume, such material was essentially not detectable in the resultant spoil pile. When inhibitory material exceeded 15% of the overburden volume it was always found in the resultant spoil pile, generally to a smaller extent, indicating partial dilution.

(12) Scatter spoiling seemed to produce the best degree of mixing, however normal spoiling could not be judged significantly different. Dump spoiling generally resulted in less mixing of inhibitory zones and should be minimized in the dragline spoiling operation.

(13) Dragline spoiling often resulted in inhibitory overburden zones being deposited at the top of the spoil piles in the potential root zone area, suggesting the need for selective placement procedures.

(14) Dragline spoiling often resulted in inhibitory zones being deposited toward the pit base. Selective placement procedures may be the means to isolate such material from a developing aquifer.

(15) Research on spoil mixing phenomena should be conducted at other mines sites which have different pit configurations and equipment for handling overburden. It is possible that given a different set of conditions the mixing/dilution of unsuitable overburden materials could be performed more effectively than that demonstrated in this study.

HYDROLOGY AND WATER QUALITY

(16) Of the three premine aquifers in the research area, the Rosebud Coal aquifer was most affected by mining. The Rosebud piezometric surface generally declined for all areas surrounding the mine. Drawdown of this aquifer was observed 457 m (1500 ft) from the pit. Decreases in overburden water levels were generally confined to areas near the mine, likely due to the poor hydraulic characteristics and discontinuous straigraphic units in overburden. The piezometric surface for the McKay Coal aquifer located below overburden and coal excavation was altered, to a limited extent, by mining. This was likely caused by blasting and overburden removal which increased leakage between the McKay Coal and other aquifers.

(17) A saturated zone at the base of the spoil materials formed and partially stabilized near the east edge of the Area B mine. Limited hydraulic data suggested the hydraulic conductivity of at least a portion of the spoil material was very slow. This may cause permanent decreases in ground-water levels downgradient from the mine. It may also produce seeps and locally increase ground-water levels in certain aquifers upgradient from the mine. During the period of study (1976-1980), ground-water flow has been into the mined area which was dewatered during mining. As the spoil ground-water levels stabilize and flow resumes downgradient through these materials, some degradation of downgradient discharge areas will likely occur.

(18) Ground-water samples from spoil material exhibited TDS values 2 to 4 times higher than other aquifers tested. The long-term impact of the mine on the areas water resources remains uncertain and it may be overshadowed by other industrial and residential development in the area. Attempts to "isolate" a complete mine such as Western Energy Company's Mine Area B would likely cause more problems than it would solve. It is probable that the best solution remains in the isolation of specific inhibitory materials within the mine by selective handling procedures.

GENERAL

(19) Overburden sample contamination from drill stem joint grease with Pb, Zn, and Cu additives caused substantial error regarding DTPA extractable trace metals. Such error could easily cause overburden zones to be mistakenly designated toxic, creating unwarranted environmental concern. A grease with teflon, Zn and Mo additives did not cause significant

sample contamination. It was recommended that petroleum base greases with teflon and talc additives (a synthetic (C_2F_4) and a mineral with no trace metal content, respectively) be used in future overburden sampling programs. However, more research is needed to clearly identify a noncontaminating grease with good lubricating performance.

(20) Overburden analysis procedures in general use throughout the West provide no indication of the post-mine ground-water trace metal content. DTPA extractable trace elements in overburden materials, a test in general use to determine plant available element levels, had poor statistical relationship to post-mine ground-water quality. Overburden columns leached with water in the laboratory produced leachates with comparable trace metal levels found in ground waters of overburden and spoil materials. Column leaching techniques are slow, expensive, and may not be practical for premine studies. An investigation should be initiated to determine whether a water saturated paste extract, which is performed on a routine basis, could provide similar trace metal levels as those attained from column leachates.

(21) Chemical differences between newly backfilled spoils and that in the premine overburden were due largely to physical mixing/dilution of materials, rather than to chemical transformations. Once the spoils were in place, the chemical status of materials did not change notably for 15 months regarding the parameters pH, EC, Pb, Ni, and Zn. Clearly toxic zones were not being generated in the spoil materials due to weathering or chemical transformation of material in its disturbed new environment.

XIII

APPENDICES

APPENDIX A

LOCATIONS, MONITORED ZONES AND DEPTH TO WATER MEASUREMENTS FOR WELLS AND PIEZOMETERS CONSTRUCTED AT THE RESEARCH SITE, WESTERN ENERGY COMPANY MINE AREA B, NEAR COLSTRIP, MT.

Table 28. Location, elevation, total depth and depth to water of wells constructed at the research site, Western Energy Company Mine Area B, near Colstrip, Montana.

Well Number	Location	Monitored Zone (3)	Well Depth (ft)	Altitude of Measuring Point (ft)(1)	1977															
					April		May		June		July		August		September		October		December	
					Date	DTW	Date	DTW	Date	DTW	Date	DTW	Date	DTW	Date	DTW	Date	DTW	Date	DTW
4-O	N47,079;E54,433	RO	19	3281.34	4/30	Dry	5/16	Dry	6/29	Dry	7/21	Dry	8/30	Dry	9/23	Dry	10/20	Dry	12/5	Dry
4-R	N47,064;E54,433	RC	43	3282.25	4/30	30.42	5/16	30.58	6/29	31.14	7/21	31.47	8/30	32.12	9/23	33.04	10/20	33.30	12/5	33.98
4-M	N47,049;E54,433	MC	82	3282.62	4/30	38.36	5/16	37.74	6/29	37.78	7/21	38.45	8/30	37.97	9/23	41.35	10/20	38.83	12/5	40.55
5-O	N46,880;E55,514	RO	47	3290.52	4/30	22.24	5/19	22.38	6/29	22.50	7/21	22.62	8/30	23.05	9/23	23.47	10/20	23.96	12/5	Destroyed
5-R	N46,860;E55,529	RC	71	3291.10	4/30	42.23	5/19	43.84	6/29	45.83	7/21	47.51	8/30	48.49	9/23	51.53	10/20	51.51	12/5	Destroyed
7-O	N45,754;E56,472	RO	79	3316.44	4/30	52.14	Buried	-	-	-	-	-	-	-	-	-	-	-	-	-
7-R	N45,739;E56,472	RC	106	3316.59	4/30	52.83	Buried	-	-	-	-	-	-	-	-	-	-	-	-	-
7-M	N45,724;E56,472	MC	132	3317.27	4/30	78.32	Buried	-	-	-	-	-	-	-	-	-	-	-	-	-
10-R	N44,474;E59,973	RC	177	3401.70	4/30	139.88	5/17	139.38	6/28	140.53	7/21	140.50	8/30	140.40	9/23	140.36	10/20	140.37	12/5	140.55
15-O	N43,610;E56,295	RO	98	3355.90	4/30	*79.72	5/17	45.55	6/28	44.85	7/23	44.52	8/26	45.27	9/23	44.80	10/21	44.68	12/5	44.69
15-R	N43,595;E56,295	RC	122	3356.15	4/30	86.67	5/17	86.70	6/28	87.25	7/23	87.35	8/26	87.60	9/23	87.82	10/21	88.40	12/5	88.90
15-M	N43,580;E56,295	MC	158	3356.96	4/30	111.32	5/17	112.71	6/28	113.90	7/23	114.07	8/26	114.05	9/23	114.43	10/21	115.20	12/5	114.90
28-O	N46,485;E56,935	RO	70	3308.16	4/30	46.25	5/17	46.83	Destroyed		-	-	-	-	-	-	-	-	-	-
28-R	N46,470;E56,935	RC	95	3308.03	4/30	55.33	5/17	59.35	Destroyed		-	-	-	-	-	-	-	-	-	-
29-O	N45,637;E57,788	RO	66	3314.89	4/30	55.58	5/17	56.73	6/28	61.22	Destroyed		-	-	-	-	-	-	-	-
29-R	N45,628;E57,801	RC	94	3314.92	4/30	56.35	5/17	57.35	6/28	62.42	Destroyed		-	-	-	-	-	-	-	-
30-O	N44,780;E58,409	RO	73	3331.01	4/30	65.49	5/17	65.73	6/28	66.18	7/21	66.45	8/30	67.32	9/23	67.69	10/20	68.21	12/5	69.07
30-R	N44,765;E58,410	RC	98	3331.18	4/30	67.47	5/17	67.44	6/28	68.16	7/21	68.67	8/30	69.39	9/23	69.78	10/20	70.36	12/5	71.23
31-O	N45,104;E54,223	RO	91	3337.38	4/30	65.62	5/16	65.44	6/28	65.80	7/21	65.48	8/30	65.47	9/23	65.42	10/20	65.54	12/5	65.93
31-R	N45,089;E54,225	RC	116	3338.02	4/30	70.17	5/16	70.11	6/28	70.52	7/21	70.56	8/30	70.65	9/23	69.73	10/20	70.14	12/5	70.79
31-M	N45,074;E54,226	MC	150	3338.90	4/30	92.88	5/16	92.25	6/28	92.01	7/21	92.06	8/30	92.95	9/23	95.10	10/20	93.00	12/5	93.61
32-O	N47,224;E57,660	RO	55	3294.55	4/30	37.11	5/18	37.26	6/29	37.50	7/22	37.61	8/26	37.90	9/23	38.08	10/20	38.24	12/5	38.53
32-R	N47,215,E57,672	RC	79	3294.51	4/30	55.31	5/18	55.41	6/29	54.62	7/22	54.90	8/26	55.83	9/23	55.12	10/20	54.85	12/5	54.95
33-O	N44,782;E57,173	RO	83	3327.81	4/30	57.34	5/19	56.71	6/29	56.78	7/21	56.59	8/30	56.97	9/23	56.81	10/20	56.93	12/5	56.92
33-R	N44,767;E57,173	RC	107	3328.15	4/30	62.36	5/19	62.69	6/28	63.28	7/21	63.43	8/30	64.13	9/23	64.71	10/20	65.76	12/5	67.31
34-R	N46,964;E55,690	RC	72	3289.40	4/30	42.71	5/19	44.12	6/29	46.58	7/21	48.66	8/30	Destroyed		-	-	-	-	-
41-O	N45,838;E51,786	RO	80	3316.28																
41-R	N45,871;E51,782	RC	110	3315.07																
41-M	N45,890;E51,779	MC	140	3314.45																
42-O	N44,222;E50,210(+)	RO	96	3327.48																
42-R	N44,206;E50,204(+)	RC	116	3328.03																
42-M	N44,192;E50,198(+)	MC	144	3328.38																
43-O	N44,112;E52,871(+)	RO	125	3349.91																
43-R	N44,106;E52,856(+)	RC	147	3349.28																
43-M	N44,099;E52,840(+)	MC	170	3348.91																
CM-1	N46,993;E57,224	MC	100	3298.93																
CR-2	N46,998;E57,247	SP	78	3299.38																
D-2	N46,124;E58,152	SP	103	3309.11																
S-27(2)	N43,895;E58,890(2)	MC	149	3351.50	4/30	113.30	5/17	113.67	6/28	114.23	7/21	113.58			9/23	113.82	10/20	114.78	12/5	113.99
S-28(2)	N43,870;E58,890(2)	RC	130	3351.82	4/30	86.37	5/17	86.57	6/28	86.96	7/21	84.44			9/23	87.43	10/20	87.56	12/5	88.44
P-06(2)	N49,675;E53,500(2)	MC	50	3275.5	4/30	22.22	5/19	22.43			7/21	23.68			9/23	24.20	10/20	24.09	12/5	24.21

* Drilling fluid had not yet drained from hole

+ Location information courtesy of Western Energy Company

(1) Top of casing elevation accurate to nearest .1 foot (approximately 3 feet above ground surface for all wells except 7-O, 7-R and 7-M

(2) Wells installed by the Montana Bureau of Mines and Geology

(3) RO = Rosebud Overburden, RC = Rosebud Coal, MC = McKay Coal, SP = Spoils

Table 28. (continued)

Well Number	1978																							
	January		February		March		April		May		June		July		August		September		October		November		November		December
	Date	DTW	Date	DTW	Date	DTW	Date	DTW	Date	DTW	Date	DTW	Date	DTW	Date	DTW	Date	DTW	Date	DTW	Date	DTW	Date	DTW	Date	DTW
4-0	1/5	Bent	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
4-R	1/5	34.47	2/15	34.34	3/14	33.91	4/24	34.65	5/16	35.29	6/21	35.54	7/12	34.56	Destroyed		-	-	-	-	-	-	-	-	-	-
4-M	1/5	38.31	2/15	37.63	3/14	37.53	4/24	37.75	5/16	37.93	6/21	35.44	7/12	37.27	Destroyed		-	-	-	-	-	-	-	-	-	-
5-0	Destroyed		-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
5-R	Destroyed		-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
7-0.	Buried		-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
7-R	Buried		-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
7-M	Buried		-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
10-R	1/5	7120.00	2/14	140.81	3/14	140.66	4/24	141.05			6/21	141.40	7/13	141.26	8/22	141.40	9/28	141.71			11/17	141.97	11/29	142.03		
15-0	1/5	45.00	2/10	45.02	3/14	45.00	4/25	43.15			6/21	45.32	7/12	45.19	8/22	45.89	9/29	45.48	10/1	50.87*			11/30	44.85		
15-R	1/5	89.38	2/10	90.06	3/14	90.52	4/25	91.20			6/21	91.34	7/12	91.69	8/22	92.56	9/29	93.50	10/1	93.55			11/30	94.17	12/1	94.23
15-M	1/5	114.55	2/10	114.25	3/14	113.92	4/25	114.85			6/21	114.39	7/11	114.29	8/22	115.78	9/29	116.59	10/1	116.76			11/30	118.04	12/1	117.85
28-0	-	-	-	-	-	-	-	-			-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
28-R	-	-	-	-	-	-	-	-			-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
29-0	-	-	-	-	-	-	-	-			-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
29-R	-	-	-	-	-	-	-	-			-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
30-0	1/5	69.61	2/14	70.22	3/14	70.56	4/24	68.70			6/21	66.87	7/13	66.87	8/22	68.32	9/29	69.51			11/17	70.58	11/29	70.86		
30-R	1/5	71.76	2/14	72.27	3/14	72.58	4/24	71.55			6/21	71.13	7/13	71.62	8/22	72.41	9/28	73.06			11/17	73.70	11/29	74.08		
31-0	1/5	65.44	2/14	65.88	3/14	65.63	4/24	66.20			6/21	66.19	7/12	66.29	8/23	66.30	9/29	66.42			11/17	66.14	11/29	66.31		
31-R	1/5	70.70	2/14	71.29	3/14	70.95	4/24	71.60			6/21	71.98	7/12	71.97	8/23	72.37	9/29	72.68			11/17	73.10	11/29	73.03		
31-M	1/5	93.45	2/14	92.88	3/14	92.92	4/24	93.90			6/21	92.53	7/12	93.77	8/23	96.05	9/29	95.79			11/17	97.26	11/29	97.10		
32-0	1/5	38.72	2/14	38.86	3/13	38.91	4/25	37.55			6/21	35.55	7/13	34.30	8/23	33.69	9/29	33.44	10/1	33.41			11/30	33.86	12/1	33.88
32-R	1/5	54.49	2/14	54.78	3/13	54.80	4/25	51.98			6/21	48.76	7/13	48.63	8/23	48.61	9/29	49.27	10/1	49.28			11/30	49.44	12/1	49.48
33-0	1/5	57.69	2/14	57.95	3/14	57.51	4/24	51.58			6/21	Buried	7/13	48.75	8/23	52.08	9/28	53.70			11/17	54.32	11/30	54.66	12/19	54.96
33-R	1/5	68.32	2/14	69.83	3/14	70.09	4/24	71.29					7/11	73.70	8/23	75.05	9/29	77.80			11/17	79.58	11/30	79.73	12/19	80.34
34-R	Destroyed		-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
41-0																			10/10	47.08	11/17	46.48	11/30	46.38		
41-R																			10/10	47.74	11/17	-	11/30	47.55		
41-M																			10/10	≈60.	11/17	47.64	11/30	50.17		
42-0																			10/10	46.26	11/17	46.21				
42-R																			10/10	47.03	11/17	47.00				
42-M																			10/10	50.17	11/17	50.33				
43-0																			10/10	73.27	11/17	73.18				
43-R	Well seal leakage observed during pumping, well redrilled on 4/6/79																		10/10	72.61	11/17	72.55				
43-M	Well seal doubtful as bore caved before grouting was completed																		10/10	80.53	11/17	79.57				
CM-1																			10/10	53.45	11/17	58.73	11/30	58.82		
CR-2																			10/10	61.36	11/17	61.03	11/30	55.21		
D-2			2/15	74.90	3/14	74.84	4/24	74.01			6/21	73.06	7/14	72.75	8/22	71.92	9/28	71.44			11/17	70.83				
S-27	1/5	113.81	2/14	113.45	3/14	113.45	3/14	114.12			6/21	114.13	7/12	112.97	8/23	114.51	9/29	116.25					11/30	117.00		
S-28	1/5	87.87	2/14	88.65	3/14	88.66	4/24	89.07			6/21	88.82	7/12	89.24	8/23	89.35	9/29	89.62					11/30	89.74		
P-06	1/5	24.22	2/15	23.90	3/14	21.97	4/24	22.54					7/13	22.25	8/22	23.26	9/29	22.91					11/30	23.40		

*After Pumping

Table 28. (continued)

Well Number	1979																											
	January		January		February		February		March		April		May		June		July		August		September		October		November		December	
	Date	DTW	Date	DTW	Date	DTW	Date	DTW	Date	DTW	Date	DTW	Date	DTW	Date	DTW	Date	DTW	Date	DTW	Date	DTW	Date	DTW	Date	DTW	Date	DTW
4-O	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
4-R	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
4-M	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
5-O	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
5-R	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
7-O	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
7-R	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
7-M	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
10-R	1/17	142.27			2/13	142.48			3/1	143.15	4/10	142.54	5/14	143.24	6/13	143.24	7/17	143.59	8/21	143.44	9/11	143.58	10/12	-	11/13	143.99	12/11	143.97
15-O	1/11	45.67	1/23	45.39	2/10	45.33	2/28	45.35			4/10	44.68	5/14	44.64	6/12	44.77	7/19	44.96	8/23	45.17	9/11	45.48	10/12	45.29	11/14	45.08	12/11	45.75
15-R	1/9	94.84	1/11	94.85	2/10	95.04	2/28	95.37			4/10	95.46	5/14	96.09	6/12	96.43	7/19	96.92	8/23	97.11	9/11	97.35	10/12	97.49	11/14	97.64	12/11	97.79
15-M	1/9	118.00	1/11	118.10	2/10	117.76	2/28	117.95			4/10	116.43	5/14	118.30	6/12	117.46	7/19	117.44	8/23	118.03	9/11	117.81	10/12	118.12	10/12	118.12	12/11	117.84
28-O	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
28-R	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
29-O	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
29-R	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
30-O	1/17	71.61			2/9	71.84			3/1	72.11	4/10	72.56	5/14	72.88	6/13	73.11	7/17	73.30	8/21	73.56	9/11	73.68	10/12	73.92	11/13	74.36	12/11	dry
30-R	1/17	74.64			2/9	74.82			3/1	75.08	4/10	75.28	5/14	75.52	6/13	75.68	7/17	75.95	8/21	76.23	9/11	76.48	10/12	76.66	11/13	76.72	12/11	77.05
31-O	1/17	66.46			2/10	66.50	2/28	66.14			4/10	66.17	5/14	66.59	6/13	66.26	7/18	66.49	8/21	66.36	9/11	66.35	10/12	66.34	11/13	66.37	12/11	66.24
31-R	1/17	73.18			2/10	73.46	2/28	73.34			4/10	73.33	5/14	73.97	6/13	74.16	7/18	74.51	8/21	75.29	9/11	75.32	10/12	75.63	11/13	75.84	12/11	75.04
31-M	1/17	96.24			2/10	96.62	2/28	95.56			4/10	95.60	5/14	98.45	6/13	96.81	7/18	95.91	8/21	98.56	9/11	97.11	10/12	97.86	11/13	97.25	12/11	95.83
32-O	1/9	34.59	1/15	34.20	2/9	34.54	2/28	34.63			4/10	33.53	5/14	32.86	6/12	32.36	7/17	31.99	8/23	32.32	9/11	32.59	10/12	32.95	11/14	33.32	12/11	33.63
32-R	1/9	50.08	1/15	50.08	2/9	50.25	2/28	50.18			4/10	48.93	5/14	48.55	6/12	47.67	7/17	47.95	8/23	48.32	9/11	48.77	10/12	49.26	11/14	49.91	12/11	50.23
33-O	Destroyed		-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
33-R	Destroyed		-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
34-R	-		-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
41-O	1/17	51.10			2/13	48.88			3/1	46.48	4/10	46.28	5/14	46.97	6/12	46.91	7/18	47.41	8/21	47.65	9/11	47.74	10/12	47.98	11/13	48.07	12/11	47.93
41-R	1/17	47.50			2/13	47.41			3/1	46.99	4/10	46.79	5/14	47.15	6/12	47.23	7/18	47.53	8/21	47.99	9/11	48.07	10/12	48.32	11/13	48.43	12/11	48.34
41-M	1/17	47.80			2/13	47.85			3/1	48.14	4/10	46.64	5/14	47.25	6/12	47.51	7/18	47.34	8/21	47.82	9/11	47.65	10/12	47.75	11/13	47.63	12/11	47.43
42-O	1/17	46.30			2/13	46.12					4/11	45.37	5/15	45.22	6/12	45.18	7/18	45.23	8/21	45.27	9/11	45.45	10/12	45.54	11/13	45.75	12/11	45.71
42-R	1/17	47.00			2/13	46.82					4/11	46.16	5/15	46.09	6/12	46.01	7/18	46.16	8/21	46.14	9/11	46.34	10/12	46.38	11/13	46.61	12/11	46.55
42-M	1/17	50.65			2/13	50.46					4/11	48.73	5/15	50.79	6/12	51.34	7/18	51.44	8/21	51.42	9/11	51.36	10/12	51.35	11/13	51.41	12/11	51.34
43-O	1/17	73.25			2/13	72.98			3/1	73.02	4/10	71.91	5/15	72.08	6/12	72.32	7/18	72.60	8/21	72.52	9/11	72.67	10/12	72.78	11/13	72.97	12/11	72.82
43-R	1/17	72.58			2/13	72.28			3/1	72.35	4/10	73.68	5/15	73.55	6/12	72.67	7/18	72.79	8/21	72.57	9/11	72.71	10/12	72.65	11/13	72.85	12/11	72.81
43-M	1/17	80.20			2/13	80.15			3/1	79.15	4/10	75.75	5/15	73.88	6/12	73.34	7/18	73.57	8/21	73.40	9/11	73.50	10/12	73.43	11/13	72.98	12/11	72.89
CM-1	1/17	57.02	1/23	59.01	2/8	59.15	2/28	58.73			4/10	57.01	5/13	57.78	6/13	56.06	7/17	57.75	8/21	58.09	9/11	57.86	10/12	58.15	11/13	58.23	12/11	58.34
CR-2	1/17	59.06	1/23	58.92	2/8	59.47	2/28	58.59			4/10	57.99	5/13	58.01	6/13	57.28	7/17	57.53	8/23	57.24	9/11	57.31	10/12	57.17	11/13	57.41	12/11	57.54
O-2	1/17	70.23			2/13	69.81					4/8	69.40	5/13	69.40	6/11	68.94	7/17	66.73	8/23	68.46	9/11	68.41	10/12	68.20	11/13	68.21	12/11	68.12
S-27	1/17	116.87			2/13	116.17							5/13	116.50	6/13	115.22	7/17	115.71	8/23	116.06	9/11	116.00	10/12	116.11	11/13	115.87	12/11	115.65
S-28	1/17	89.85			2/13	89.82							5/13	90.51	6/13	90.36	7/17	90.65	8/23	90.69	9/11	90.74	10/12	90.60	11/13	90.78	12/11	90.95
P-06	1/17	23.48			2/13	23.09							5/13	21.06	6/12	22.24	7/17	23.16	8/23	23.84	9/11	24.09	10/12	24.33	11/13	24.14	12/11	24.02

Table 28. (continued)

Well	1980																			
	January		February		March		May		May		June		July		September		October		November		December	
Number	Date	DTW	Date	DTW	Date	DTW	Date	DTW	Date	DTW	Date	DTW	Date	DTW	Date	DTW	Date	DTW	Date	DTW	Date	DTW
4-0	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
4-R	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
4-M	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
5-0	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
5-R	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
7-0	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
7-R	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
7-M	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
10-R	1/19	144.35	2/27	144.10	3/24	144.17	5/7	144.42	5/28	144.23	6/27	144.22	7/21	144.53	9/4	144.49	10/5	144.84	11/6	144.58	12/9	144.64
15-0	1/19	45.34	2/27	45.24	3/24	46.36	5/7	46.07	5/27	45.66	6/27	46.27	7/21	46.24	9/4	45.63	10/5	46.19	11/6	46.64	12/9	46.77
15-R	1/19	98.18	2/27	97.77	3/24	97.94	5/7	99.21	5/27	99.24	6/27	99.32	7/21	99.71	9/4	99.75	10/5	100.02	11/6	99.65	12/9	100.10
15-M	1/19	117.63	2/27	117.43	3/24	118.31	5/7	118.78	5/27	118.51	6/27	118.25	7/21	118.20	9/4	118.33	10/5	118.53	11/6	118.40	12/9	120.14
28-0	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
28-R	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
29-0	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
29-R	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
30-0	1/19	Dry	2/27	Dry	3/24	Dry	5/7	Dry	5/27	Dry	6/27	Dry	7/21	Dry	9/4	Dry	10/5	Dry	11/6	Dry	12/9	Dry
30-R	1/19	77/46	2/27	77.80	3/24	78.11	5/7	78.48	5/27	78.55	6/27	78.86	7/21	78.96	9/4	79.30	10/5	79.46	11/6	79.55	12/9	79.57
31-0	1/19	66.67	2/27	66.22	3/24	66.55	5/7	66.88	5/28	67.15	6/26	66.99	7/21	67.69	9/4	67.99	10/5	68.37	11/6	68.51	12/9	68.98
31-R	1/19	76.41	2/27	76.61	3/24	77.29	5/7	79.33	5/28	81.11	6/26	81.35	7/21	81.78	9/4	82.20	10/5	82.60	11/6	84.31	12/9	85.65
31-M	1/19	96.37	2/27	96.78	3/24	97.09	5/7	97.03	5/28	97.19	6/26	96.55	7/21	96.84	9/4	96.71	10/5	97.47	11/6	96.26	12/9	97.69
32-0	1/19	34.14	2/27	34.56	3/24	34.89	5/7	35.44	5/27	35.65	-	-	7/21	36.23	9/4	36.86	10/5	37.20	11/6	37.59	12/9	37.96
32-R	1/19	50.96	2/27	50.95	3/24	51.33	5/7	51.79	5/27	51.85	-	-	7/21	52.20	9/4	52.55	10/5	52.72	11/6	52.64	12/9	52.93
33-0	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
33-R	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
34-R	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
41-0	1/19	48.17	2/27	48.14	3/24	48.20	5/7	48.53	5/28	48.62	6/26	48.82	7/21	49.17	9/4	49.31	10/5	49.02	11/6	49.57	12/9	49.68
41-R	1/19	48.61	2/27	48.56	3/24	48.52	5/7	48.85	5/28	48.89	6/26	48.90	7/21	49.25	9/4	49.54	10/5	49.18	11/6	49.72	12/9	49.90
41-M	1/19	47.43	2/27	47.43	3/24	47.55	5/7	47.82	5/28	48.68	6/26	47.69	7/21	47.93	9/4	48.34	10/5	47.78	11/6	48.32	12/9	48.40
42-0	1/19	46.04	2/27	45.79	3/24	45.97	5/7	46.12	5/28	46.10	6/26	45.98	7/21	46.41	9/4	46.59	10/5	46.34	11/6	46.81	12/9	47.07
42-R	1/19	46.88	2/27	46.60	3/24	46.74	5/7	46.88	5/28	46.83	6/26	46.74	7/21	47.17	9/4	47.28	10/5	47.02	11/6	47.50	12/9	47.75
42-M	1/19	51.29	2/27	51.18	3/24	51.22	5/7	51.36	5/28	51.26	6/26	51.22	7/21	51.43	9/4	51.55	10/5	51.62	11/6	51.74	12/9	51.87
43-0	1/19	73.48	2/27	72.87	3/24	73.10	5/7	73.46	5/28	73.25	6/26	73.18	7/21	73.91	9/4	74.06	10/5	74.37	11/6	74.00	12/9	74.57
43-R	1/19	73.33	2/27	73.03	3/24	73.21	5/7	73.75	5/28	73.72	6/26	73.94	7/21	74.41	9/4	74.55	10/5	74.88	11/6	75.03	12/9	75.59
43-M	1/19	73.31	2/27	73.03	3/24	73.17	5/7	73.63	5/28	73.56	6/26	74.19	7/21	74.37	9/4	74.58	10/5	74.77	11/6	74.86	12/9	75.40
CM-1	1/19	59.03	2/28	*56.79	3/24	59.94	5/7	59.91	5/27	59.78	6/27	60.84	7/21	62.57	9/5	63.43	10/5	61.94	11/6	60.90	12/9	60.74
CR-2	1/19	57.98	2/28	57.50	3/24	57.56	5/7	57.76	5/27	57.69	6/27	57.63	7/21	57.83	9/5	57.91	10/5	57.94	11/6	57.58	12/9	58.19
D-2	1/20	68.08	2/28	67.90	3/24	67.85	5/7	67.88	5/27	67.77	6/27	67.80	7/21	67.79	9/5	67.70	10/5	67.81	11/6	67.50	12/9	67.78
S-27	1/19	115.85	2/27	115.94	3/24	116.68	5/7	116.78	5/28	116.75	6/27	116.55	7/21	117.26	9/5	117.72	10/5	117.94	11/6	117.02	12/9	116.80
S-28	1/19	91.22	2/27	90.58	3/24	90.95	5/7	90.96	5/28	90.93	6/27	90.86	7/21	90.87	9/5	90.98	10/5	91.17	11/6	90.59	12/9	90.84
P-06	1/19	24.06	2/28	24.08	3/24	24.01	5/7	24.12	5/29	24.28	6/27	24.36	7/21	24.74	9/5	25.13	10/5	25.24	11/6	25.05	12/9	25.04

Table 29. Location, elevation, total depth and depth to water piezometers constructed at the research site, Western Energy Company Mine Area B, near Colstrip, MT.

Piezometer Number	Location	Monitored Zone(3)	Piezometer Depth	Altitude of Measuring Point(ft)	1977															
					April		May		June		July		August		September		November		December	
					Date	DTW	Date	DTW	Date	DTW	Date	DTW	Date	DTW	Date	DTW	Date	DTW	Date	DTW
N-1	N47,216;E55,770	RO		3289,58	4/30	21.28	5/19	21.51	Destroyed		-	-	-	-	-	-	-	-	-	-
N-2	N46,590;E56,300	RO		3320.93	4/30	53.47	5/19	53.87	6/29	54.81	7/21	55.66	Destroyed		-	-	-	-	-	-
N-3	N47,460;E55,200	RO		3277.39	4/30	10.43	5/19	10.63	6/29	10.56	7/21	10.72	8/17	11.01	Destroyed		-	-	-	-
S-1	N45,730;E57,400	RO		3311.90	4/30	50.86	5/19	51.46	6/29	52.98			8/30	58.85	Destroyed		-	-	-	-
S-2	N45,200;E57,920	RO		3322.59	4/30	58.67	5/19	59.01	6/28	59.79	Destroyed		-	-	-	-	-	-	-	-
T-1A-P	N45,051;E59,215	SO		3362.63													11/11	126.90	12/5	126.76
T-1Q-P	N45,697;E58,682	SO	140	3352.85													11/11	-	12/5	121.29
T-2A-P	N46,537;E57,651	SO	68.5	3288.82													11/11	61.70	12/5	61.03
T-2Q-P	N47,001;E57,000	SD	80	3302.38													11/11	76.81	12/5	76.56
T-3A-P	N47,487;E56,788	SO		3299.33													11/11	62.08	12/5	61.86
T-3Q-P	N47,017;E57,436	RC(1)	79	3295.34													11/11	54.94	12/5	56.58
T-4A-P	N47,800;E55,921	SP(2)		3307.11													11/11	77.09	12/5	78.01
T-4M-P	N47,803;E56,519	RC(1)		3311.97													11/11	68.95	12/5	69.35
T-5B-P	N48,203;E55,514	SP	65	3288.94													11/15	59.10	12/5	59.18
T-5K-P	N48,196;E55,961	SP	70	3307.53													11/15	67.98	12/5	68.68
D-1	N45,694;E58,450	SP	115	3326.14													11/11	97.01	12/5	96.87
CP-1	N47,107;E57,173	SP	83	3299.45																
CP-2	N46.899;E57,048	SP	75.5	3298.19																
CP-3	N46,902;E57,407	SP	72	3292.82																
1+00	N47.052;E55.855	SP	63	3285.16																
6+00	N47.261;E55.526	SP		3269.20																

Piezometer Number	1978																							
	January		February		March		April		May		June		July		August		September		October		November		December	
	Date	DTW	Date	DTW	Date	DTW	Date	DTW	Date	DTW	Date	DTW	Date	DTW	Date	DTW	Date	DTW	Date	DTW	Date	DTW	Date	DTW
N-1	Destroyed		-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
N-2	Destroyed		-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
N-3	Destroyed		-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
S-1	Destroyed		-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
S-2	Destroyed		-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
T-1A-P	1/5	125.73	2/14	126.01	3/14	125.23	4/24	125.64			6/21	124.09			8/22	124.15	9/29	124.16			11/17	123.56	12/19	123.27
T-1Q-P	1/5	120.54	2/15	120.19	3/14	119.83	4/24	119.50			6/21	118.44	7/12	118.30	8/22	117.58	9/29	117.12			-	-	-	-
T-2A-P	1/5	60.08	2.15	59.75	3/14	59.47	4/24	59.11			6/21	57.32	7/13	56.65	8/22	55.74	9/29	55.13			-	-	-	-
T-2Q-P	1/5	75.70	2/15	75.14	3/14	74.71	4/24	74.38			6/21	-	7/13	72.54	8/22	71.56	9/29	71.01			11/17	70.01	12/19	69.49
T-3A-P	1/5	61.34	2/15	61.47	3/14	61.20	4/24	59.52			6/21	54.62	7/13	53.61	8/22	52.37	9/29	53.24			11/17	53.62	12/19	54.06
T-3Q-P	1/5	55.99	2/15	56.43	3/14	56.22	4/24	54.31			6/21	-	7/13	50.26	8/22	49.68	9/29	49.71			11/17	49.65	12/19	49.85
T-4A-P	1/5	77.36	2/15	77.24	3/14	72.02	4/24	76.62			6/21	73.73	7/13	73.73	8/22	73.43	9/29	72.92			11/17	72.40	12/19	72.02
T-4M-P	1/5	69.95	2/15	70.93	3/14	70.88	4/24	66.40			6/21	63.13	7/13	62.94	8/22	62.53	9/29	62.57			11/17	62.92	12/19	63.24
T-5B-P	1/5	58.82	2/15	58.53	3/14	58.32	4/24	57.82			6/21	54.28	7/13	54.49	8/22	54.53	9/29	53.88			11/17	53.57	12/19	53.27
T-5K-P	1/5	67.50	2/15	68.53	3/14	68.85	4/24	65.90			6/21	62.44	7/13	62.06	8/22	61.60	9/29	61.32			Casing Broken		-	-
D-1	1/5	95.72	2/15	95.43	3/14	95.18	4/24	94.89			6/21	93.69	7/12	93.64	8/22	92.42	9/29	92.08			11/17	91.29	12/19	91.01
CP-1																			10/10	53.00	11/17	53.13	12/19	53.33
CP-2																			10/10	66.19	11/17	65.35	12/19	64.84
CP-3																			10/10	50.29	11/17	50.00	12/19	49.85
1+00																			10/10	56.92	11/22	56.31	12/19	55.90
6+00																			10/10	37.94	11/22	37.10	12/19	36.97

Piezometer Number	1979																							
	January		February		March		April		May		June		July		August		September		October		November		December	
	Date	DTW	Date	DTW	Date	DTW	Date	DTW	Date	DTW	Date	DTW	Date	DTW	Date	DTW	Date	DTW	Date	DTW	Date	DTW	Date	DTW
N-1	Destroyed		-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
N-2	Destroyed		-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
N-3	Destroyed		-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
S-1	Destroyed		-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
S-2	Destroyed		-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
T-1A-P	1/17	123.10	2/13	123.07			4/18	122.44	5/13	122.94	6/13	122.50	7/17	122.72	8/23	122.39	9/11	122.40	10/12	122.07	11/13	122.38	12/11	122.31
T-1Q-P	1/17	115.99							5/13	115.05	6/11	114.84	7/17	114.60	-	-	9/11	114.24	10/12	114.04	11/13	114.06	12/11	113.95
T-2A-P	Destroyed		-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
T-2Q-P	1/17	69.30	2/9	68.80			4/18	67.75	5/13	67.63	6/11	67.14	7/17	66.85	8/23	66.36	9/12	66.15	10/12	65.82	11/13	65.70	12/11	65.66
T-3A-P	1/17	53.06	2/13	53.96			4/18	52.02	5/13	52.26	6/11	51.09	7/17	51.66	8/23	52.38	9/11	52.87	10/12	53.69	11/13	54.52	12/11	55.14
T-3Q-P	Destroyed		-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
T-4A-P	1/17	71.81	2/13	71.54			Filled with sand						-	-	-	-	-	-	-	-	-	-	-	-
T-4M-P	1/17	63.30	2/13	63.23			4/18	62.27	5/13	61.77	6/11	61.15	7/17	61.71	8/23	62.46	9/11	62.89	10/12	63.73	11/13	64.57	12/11	65.20
T-5B-P	1/17	53.04	2/13	52.77			4/18	51.50	5/13	51.24	6/11	51.19	7/17	51.02	8/23	50.76	9/11	50.80	10/12	50.64	11/13	50.60	12/11	50.59
T-5K-P	Casing Broken		-	-					5/13	60.42	6/11	60.02	7/17	60.23	8/23	60.56	9/11	60.90	10/12	61.31	11/13	62.18	12/11	62.63
D-1	1/17	90.71	2/13	90.31			4/18	90.03	5/13	89.95	6/11	89.34	7/17	89.34	8/23	89.07	9/11	89.01	10/12	88.72	11/13	88.75	12/11	88.75
CP-1	1/17	53.55	2/13	53.31			4/18	52.22	5/13	51.86	6/11	50.97	7/17	51.28	8/23	51.73	9/11	51.98	10/12	52.48	11/13	53.19	12/11	53.66
CP-2	1/17	64.46	2/9	64.00			4/18	62.85	5/13	62.65	6/11	62.15	7/17	61.70	8/23	61.14	9/11	61.09	10/12	60.16	11/13	60.51	12/11	60.41
CP-3	1/17	50.00	2/9	49.82			4/18	48.84	5/13	48.89	6/13	48.19	7/17	48.49	8/23	48.06	9/11	48.67	10/12	48.55	11/13	49.16	12/11	49.43
1+00	1/17	55.58	2/13	55.10					5/21	53.70	6/11	53.17	7/17	52.92	8/23	52.44	9/11	52.22	10/12	51.79	11/13	51.55	12/11	51.28
6+00	1/17	37.00	2/13	36.54					5/21	35.77	6/11	35.34	7/17	35.00	8/23	34.92	9/11	34.65	10/12	34.35	11/13	34.15	12/11	33.89

(1) Rosebud Coal and Overburden
(2) Coal Fender
(3) RO = Rosebud Overburden, RC = Rosebud Coal, MC = McKay Coal, SP = Spoils

Table 29. (continued)

Piezometer Number	Amount of Original Casing Removed (added to readings in table)	1980																					
		January		February		March		May		May		June		July		September		October		November		December	
		Date	DTW	Date	DTW	Date	DTW	Date	DTW	Date	DTW	Date	DTW	Date	DTW	Date	DTW	Date	DTW	Date	DTW	Date	DTW
N-1		-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
N-2		-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
N-3		-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
S-1		-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
S-2		-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
T-1A-P	2.59	1/19	122.70	2/27	121.80	3/24	122.07	5/7	122.05	5/28	121.83	6/27	121.49	7/21	121.92	9/5	121.94	10/5	122.13	11/6	121.62	12/9	121.98
T-1Q-P	15.63	1/19	114.08	2/28	113.71	3/24	113.63	5/7	113.65	5/27	113.38	6/27	113.35	7/21	113.32	9/5	113.33	10/5	113.35	11/6	112.81	12/9	113.42
T-2A-P	2.06	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
T-2Q-P	2.54	1/19	65.74	2/28	65.11	3/24	65.00	5/7	64.97	5/27	64.79	6/27	64.60	7/21	64.62	9/5	64.48	10/5	64.59	11/6	64.70	12/9	64.57
T-3A-P	2.62	1/19	56.03	2/28	56.39	3/24	56.79	5/7	57.34	5/27	57.41	6/27	57.79	7/21	58.22	9/5	58.66	10/5	58.75	11/6	58.60	12/9	59.04
T-3Q-P	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
T-4A-P	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
T-4M-P	2.00	1/19	66.07	2/28	66.60	3/24	66.96	5/7	67.62	5/27	67.75	6/27	68.12	7/21	68.37	9/5	68.86	10/5	69.04	11/6	69.23	12/9	69.63
T-5B-P	-	1/19	50.68	2/28	50.23	3/24	50.10	5/7	49.98	5/27	49.92	6/27	49.71	7/21	49.61	9/5	49.41	10/5	49.58	11/6	49.34	12/9	49.57
T-5K-P	1.94	1/19	63.55	2/28	63.60	3/24	64.03	5/7	64.59	5/27	64.72	6/27	64.67	7/21	65.30	9/5	65.59	10/5	65.59	11/6	65.71	12/9	66.35
D-1	8.45	1/19	88.78	2/28	88.44	3/24	88.39	5/7	88.28	5/27	88.13	6/27	88.11	7/21	88.17	9/5	88.17	10/5	88.34	11/6	87.89	12/9	88.03
CP-1	2.79	1/19	54.45	2/28	54.45	3/24	54.74	5/7	55.13	5/27	55.24	6/27	55.45	7/21	55.66	9/5	55.82	10/5	56.20	11/6	55.95	12/9	56.55
CP-2	2.89	1/19	-	2/28	59.70	3/24	59.52	5/7	59.34	5/27	59.18	6/27	59.11	7/21	59.02	9/5*	58.72	10/5	58.67	11/6	58.33	12/9	58.56
CP-3	2.10	1/20	49.67	2/28	49.72	3/24	49.88	5/7	50.22	5/27	49.93	6/27	50.09	7/21	50.26	9/5	50.60	10/5	50.34	11/6	50.42	12/9	Destoryed
1+00	-	1/19	51.03	2/28	50.31	3/24	49.94	5/7	49.62	5/27	49.36	6/27	49.02	7/21	48.77	9/5	48.17	10/5	47.98	11/6	47.99	12/9	48.09
6+00	-	Destroye																					

*Ponded water may have entered piezometer

APPENDIX B

WATER QUALITY ANALYSES FOR ALL WATER SAMPLES TAKEN WITHIN THE RESEARCH SITE, WESTERN ENERGY COMPANY MINE AREA B, NEAR COLSTRIP, MT.

Table 30. Chemical analyses showing major constituents in surface and ground water collected from Area B of the Western Energy Mine near Colstrip, Montana.

Well Number	Date Sampled	Specific (field) Conductance µmhos/cm	Specific (lab) Conductance µmhos/cm	pH (field)	pH (lab)	Total Dissolved Solids (Calc.)	Alkalinity as ($CaCO_3$)	Hardness as ($CaCO_3$)	Calcium (Ca)	Magnesium (Mg)	Sodium (Na)	Sulfate (SO_4)	Bicarbonate (HCO_3)	Carbonate (CO_3)
						milligrams per liter								
E.F. Armells	5-19-77	1310	1640	7.8	8.3	1314	20	789	128	114	73	979	24	.0
E.F. Armells	3-10-78	95		8.2	8.2	89	65	48	10	6	3	30	79	.0
E.F. Armells	3-11-78	127		8.0	8.1	102	52	65	14	7	4	44	64	.0
E.F. Armells	3-13-78		<150		8.1	90	48	64	13	7	4	34	58	.0
E.F. Armells	3-17-78		<150		8.2	455	146	298	51	41	38	232	178	.0
E.F. Armells	4-18-78		2700		8.3	2724	529	1641	228	260	184	1716	645	.0
E.F. Armells	5-08-78	1200	1000	7.6	8.5	964	201	594	92	89	56	609	209	9.0
E.F. Armells	5-19-78	1050	960	7.6	8.5	991	178	584	84	91	59	649	192	6.0
E.F. Armells	5-19-78	1330	1350	7.6	8.5	1090	90	775	105	124	67	737	79	7.5
E.F. Armells	7-13-78	2740	2400	8.1	8.3	2237	496	1257	198	185	176	1364	605	.0
E.F. Armells	8-22-78	2760	2550	8.2	8.3	2541	438	1841	162	349	232	1514	534	.0
E.F. Armells	4-10-79	2060	2100	8.2	8.2	1990	523	1349	172	223	149	1109	638	.0
E.F. Armells	5-14-79	1950	1970	7.7	7.8	1926	455	1173	156	190	145	1137	555	.0
E.F. Armells	6-12-79	2410	2200	7.8	8.2	2267	490	1284	198	191	167	1391	598	.0
E.F. Armells	7-16-79	2860	2080	8.1	8.3	2691	615	1424	195	225	177	1682	750	.0
E.F. Armells	8-23-79	2860	2950	8.4	8.5	2423	425	1231	79	251	248	1555	458	15.0
Impoundment	7-13-78	1190	1000	8.3	8.7	836	175	528	81	79	86	485	183	7.5
Impoundment	8-23-78	1340	1200	8.0	8.5	999	142	696	109	103	82	619	150	6.0
Impoundment	9-29-78	1260	1150	7.9	8.3	969	108	606	81	98	68	651	131	.0
Impoundment	3-2-79	160	300	6.6	8.0	163	58	114	26	12	9	79	70	.0
Impoundment	4-10-79	300	300	8.2	8.2	210	70	167	34	20	16	94	85	.0
Impoundment	5-14-79	1100	1100	8.4	8.4	1013	238	613	84	98	86	602	265	6.0
Impoundment	6-12-79	1580	1250	8.6	8.5	1257	232	730	102	116	99	803	247	9.0
Impoundment	7-17-79	1920	1400	9.1	8.6	1612	137	801	70	152	147	1148	143	6.0
4-R	4-20-77	1470	1800	7.6	7.6	1292	463	861	152	117	79	656	564	.0
4-R	7-25-77	1310	1350	7.2	7.9	1238	473	907	165	120	86	566	576	.0
4-R	12-15-77	1330	1600	7.5	7.9	1247	475	815	153	105	132	566	580	.0

(continued on following page)

Table 30. (continued)

Well Number	Date Sampled	Specific (field) Conductance µmhos/cm	Specific (lab) Conductance µmhos/cm	pH (field)	pH (lab)	Total Dissolved Solids (Calc.)	Alkalinity as ($CaCO_3$)	Hardness as ($CaCO_3$)	Calcium (Ca)	Magnesium (Mg)	Sodium (Na)	Sulfate (SO_4)	Bicarbonate (HCO_3)	Carbonate (CO_3)
						milligrams per liter								
4-R	5-16-78	1370	1290	7.0	8.7	1254	515	792	134	111	80	641	506	30.0
4-R	6-22-78	1551	1500	7.2	8.3	1306	474	923	166	123	99	624	578	.0
4-R	7-12-78	1470	1400	7.2	8.2	1198	455	799	140	109	96	570	555	.0
4-R	8-17-78	1560	1900	7.3	8.2	1207	472	868	153	118	100	543	575	.0
4-M	4-20-77	1520	1800	7.8	7.8	1269	425	674	122	90	142	656	518	.0
4-M	7-25-77	1610	1700	7.5	8.0	1472	455	910	166	120	175	717	555	.0
4-M	12-15-77	1460	1650	7.4	7.9	1423	488	815	153	105	168	699	595	.0
4-M	5-16-78	1510	1310	7.3	8.3	1316	416	611	116	78	160	706	508	.0
4-M	6-22-78	1550	1600	7.3	8.3	1248	495	818	154	105	189	496	604	.0
4-M	7-12-78	1680	1640	7.4	8.3	1205	478	701	134	89	184	505	513	.0
4-M	8-17-78	1700	1700	7.2	8.3	1319	494	773	148	98	184	589	602	.0
5-R	4-28-77	2610	2700	7.0	7.5	2563	685	1660	195	285	160	1396	836	.0
5-R	7-22-77	3100	2600	7.1	7.8	2988	738	2107	225	375	165	1693	900	.0
7-0	4-27-77	1710	1800	7.3	7.6	1607	548	1144	120	205	87	803	668	.0
7-R	4-27-77	1330	1380	7.6	7.7	1134	285	680	122	91	107	636	348	.0
7-M	4-27-77	1720	1600	7.7	8.0	1512	330	743	117	109	170	912	408	.0
10-R	7-24-77	2760	2600	7.1	7.9	2552	893	1555	226	240	252	1275	1089	.0
10-R	7-13-78	2140	1900	7.0	8.4	1564	775	886	140	130	240	593	897	.0
10-R	8-22-78	2090	2000	7.2	8.3	1675	749	910	140	136	248	696	914	.0
10-R	9-28-78	1990	1800	7.0	8.2	1546	690	908	149	130	226	620	842	.0
10-R	3-1-79	1550	2000	7.2	8.0	1434	726	625	95	94	209	595	884	.0
10-R	4-10-79	1630	1850	7.2	7.5	1577	712	827	136	118	236	643	868	.0
10-R	5-14-79	-	1600	7.1	7.2	1535	730	759	127	107	251	597	891	.0
10-R	6-13-79	1900	1800	7.0	8.2	1733	720	750	146	94	249	800	878	.0
10-R	7-18-79	2180	1550	7.1	8.0	1696	728	767	155	92	248	744	888	.0
10-R	8-21-79	1700	1600	7.1	7.0	1674	772	746	124	106	247	716	941	.0
10-R	11-14-79	2900	2820	7.3	8.0	1948	863	1025	146	160	323	784	1052	.0

(continued on following page)

Table 30. (continued)

Well Number	Date Sampled	Specific (field) Conductance µmhos/cm	Specific (lab) Conductance µmhos/cm	pH (field)	pH (lab)	Total Dissolved Solids (Calc.)	Alkalinity as ($CaCO_3$)	Hardness as ($CaCO_3$)	Calcium (Ca)	Magnesium (Mg)	Sodium (Na)	Sulfate (SO_4)	Bicarbonate (HCO_3)	Carbonate (CO_3)
						milligrams per liter								
10-R	2-27-80	2600	2400	6.9	7.5	2122	917	1258	182	195	272	900	1118	.0
10-R	5-28-80	2700	2340	7.1	7.3	2413	901	1607	216	259	270	1107	1098	.0
10-R	9-4-80	1900	2890	7.0	7.3	2495	926	1545	231	235	294	1157	1129	.0
10-R	12-10-80	1600	2160	7.1	7.0	2273	943	1241	198	181	243	1066	1150	.0
15-0	4-29-77	2485	2050	9.9	9.5	2205	52+	1064	278	90	162	1653	9.2+	13.5
15-0	7-23-77	2050	2100	8.3	8.2	2264	48+	1257	212	177	183	1630	57.9+	.0
15-0	7-12-78	2460	2500	7.2	8.4	2255	540	1419	238	200	185	1305	634	6.0
15-0	8-22-78	2540	2400	7.2	8.2	2362	532	1531	250	220	192	1372	648	.0
15-0	9-29-78	2440	2300	6.8	8.2	2339	534	1575	266	221	176	1345	653	.0
15-0	11-30-78	2620	1950	7.1	8.0	2388	530	1517	228	230	359	1243	647	.0
15-0	2-28-79	2000	2600	7.1	8.2	2105	544	1107	174	163	168	1264	663	.0
15-0	4-10-79	2100	2200	7.4	7.5	2337	520	1438	251	197	178	1379	634	.0
15-0	5-14-79	-	1950	7.2	7.4	2239	524	1177	191	170	193	1351	639	.0
15-0	6-12-79	2300	2150	7.1	7.9	2274	528	1263	252	154	192	1338	644	.0
15-0	7-18-79	2330	1900	7.2	8.2	2122	535	1203	226	155	180	1215	653	.0
15-0	8-23-79	2135	1900	7.0	7.4	2073	545	1066	179	150	156	1237	665	.0
15-0	11-14-79	2700	2570	7.1	8.3	1827	545	1109	163	170	181	967	664	.0
15-0	2-27-80	2100	1710	7.1	7.5	1869	575	1211	188	180	173	960	702	.0
15-0	5-28-80	2200	2010	7.2	7.3	2027	600	1357	197	210	182	1060	732	.0
15-0	9-4-80	1900	1930	7.1	7.1	1940	532	1151	162	182	172	1086	649	.0
15-0	12-10-80	1800	1670	7.2	7.1	1883	548	1147	193	162	148	1035	668	.0
15-R	7-23-77	1470	1600	7.6	8.3	1474	433	832	135	120	167	774	528	.0
15-R	7-12-78	1630	1600	7.6	8.3	1313	500	841	142	118	117	624	610	.0
15-R	8-22-78	1750	1750	7.5	8.0	1480	538	1003	164	144	124	711	656	.0
15-R	9-29-78	1680	1600	9.9+	8.2	1415	515	1034	170	148	108	669	628	.0
15-R	11-30-78	1850	1350	7.3	8.0	1455	518	1032	151	159	201	622	631	.0
15-R	2-28-79	1340	1850	7.2	8.1	1380	538	798	120	121	93	713	656	.0
15-R	4-10-79	1480	1620	7.5	7.9	1487	535	1047	175	148	98	724	653	.0
15-R	5-14-79	1610	1550	7.3	7.4	1411	550	877	140	128	112	681	671	.0
15-R	6-12-79	1790	1500	7.2	8.1	1487	549	941	183	118	108	729	670	.0
15-R	7-18-79	1710	1410	7.4	8.1	1435	540	910	183	110	108	690	659	.0
15-R	8-23-79	1600	1410	7.1	8.1	1375	555	819	128	122	89	683	677	.0

Table 30 (continued)

Well Number	Date Sampled	Specific (field) Conductance µmhos/cm	Specific (lab) Conductance µmhos/cm	pH (field)	pH (lab)	Total Dissolved Solids (Calc.)	Alkalinity as (CaCO$_3$)	Hardness as (CaCO$_3$)	Calcium (Ca)	Magnesium (Mg)	Sodium (Na)	Sulfate (SO$_4$)	Bicarbonate (HCO$_3$)	Carbonate (CO$_3$)
						milligrams per liter								
15-R	11-14-79	2000	2080	7.2	7.1	1270	550	873	123	137	107	554	671	.0
15-R	2-27-80	1600	1580	7.3	7.4	1246	625	906	135	138	78	501	763	.0
15-R	5-28-80	1700	1670	7.2	7.4	1442	623	1048	142	168	101	639	760	.0
15-R	9-4-80	1500	1580	7.2	7.3	1388	535	957	135	151	107	664	653	.0
15-R	12-10-80	1450	1400	7.3	7.7	1420	582	889	145	128	95	686	709	.0
15-M	7-23-77	1680	1800	7.7	8.3	1756	478	1253	180	195	127	942	583	.0
15-M	7-11-78	2450	2000	7.6	8.4	1957	461	1093	190	150	251	1086	532	7.5
15-M	8-22-78	2540	2600	7.4	8.1	2228	507	1286	218	180	264	1254	618	.0
15-M	9-29-78	2590	1600	7.0	8.3	2225	185+	1356	241	183	253	1430	226+	.0
15-M	11-30-78	2750	1950	7.3	8.1	2525	518	1278	208	184	510+	1306	631	.0
15-M	2-28-79	1920	2800	-	8.0	2133	509	920	154	130	224	1312	621	.0
15-M	4-10-79	-	2400	7.3	7.8	2400	515	1285	244	164	254	1409	628	.0
15-M	5-14-79	2410	2200	7.3	7.8	2291	533	1092	203	142	275	1335	650	.0
15-M	6-12-79	2720	2400	7.2	7.9	2521	513	1208	261	135	285	1514	625	.0
15-M	7-18-79	2520	2000	7.3	7.9	2543	513	1274	271	145	284	1515	625	.0
15-M	8-23-79	2360	1950	7.1	7.6	2388	588	1044	191	137	252	1436	717	.0
15-M	11-14-79	3200	2900	7.3	8.1	2092	535	1127	182	164	330	1080	653	.0
15-M	2-28-80	2500	2160	7.0	7.5	2320	575	1279	212	182	287	1276	702	.0
15-M	5-28-80	2600	2300	7.1	7.2	2651	901	1398	212	211	271	1400	1098	.0
15-M	9-4-80	1800	2380	7.1	7.2	2331	560	1315	210	192	285	1291	683	.0
15-M	12-10-80	2250	2170	7.0	7.2	2309	558	1204	214	163	260	1324	680	.0
28-0	4-29-77	1220	1220	7.6	8.1	977	400	700	75	124	61	469	488	.0
28-R	4-29-77	1070	1020	8.9	8.6	857	95	499	54	88	77	579	104	3.0
29-0	4-27-77	525	550	7.8	8.2	345	265	322	45	51	11	77	323	.0
29-R	4-27-77	805	900	7.4	7.9	636	398	456	62	74	40	220	485	.0
30-0	4-29-77	960	900	-	8.2	655	290	521	60	90	25	304	353	.0
30-0	7-24-77	850	690	7.7	8.6	559	357	520	55	93	18	183	387	12.0
30-0	6-22-78	760	680	7.8	8.3	434	320	375	34	70	16	119	390	.0
30-0	7-13-78	730	600	7.8	8.7	335	335	430	38	70	19	117	336	18.0
30-0	8-17-78	650	650	7.5	8.4	460	328	429	40	80	23	122	381	4.5
30-0	9-29-78	700	700	7.1	8.3	437	303	462	40	88	17	109	369	.0
30-0	11-30-78	570	500	7.8	8.0	465	303	443	49	78	36	119	369	.0
30-0	2-9-79	500	650	7.2	8.0	446	330	398	43	70	18	115	403	.0
30-0	3-1-79	490	750	7.6	8.1	433	329	306	27	58	15	133	401	.0
30-0	4-10-79	-	700	-	8.3	646	330	452	44	83	18	146	403	.0
30-0	5-14-79	690	660	7.8	8.0	623	346	374	31	72	24	150	422	.0
30-0	6-13-79	700	600	7.8	8.3	512	343	400	46	69	18	167	418	.0
30-0	7-18-79	790	580	8.0	8.1	495	340	388	40	70	20	154	415	.0

Table 30. (continued)

Well Number	Date Sampled	Specific (field) Conductance µmhos/cm	Specific (lab) Conductance µmhos/cm	pH (field)	pH (lab)	Total Dissolved Solids (Calc.)	Alkalinity as $(CaCO_3)$	Hardness as $(CaCO_3)$	Calcium (Ca)	Magnesium (Mg)	Sodium (Na)	Sulfate (SO_4)	Bicarbonate (HCO_3)	Carbonate (CO_3)
						milligrams per liter								
30-R	4-29-77	920	900	7.9	8.4	653	315	509	48	94	30	290	384	.0
30-R	7-24-77	950	870	7.6	8.5	687	397	607	60	111	24	265	400	21.0
30-R	12-15-77	810	950	7.7	8.3	717	345	574	61	102	39	304	421	.0
30-R	6-23-78	760	700	7.8	8.3	434	306	389	37	72	18	122	374	.0
30-R	7-13-78	660	640	7.7	8.5	457	330	388	40	70	22	137	348	13.5
30-R	8-22-78	730	700	7.2	8.4	528	340	476	42	90	26	170	390	6.0
30-R	9-29-78	900	870	7.3	8.3	578	328	598	50	115	19	195	400	.0
30-R	11-29-78	1040	720	7.8	8.1	721	338	663	51	130	40	294	412	.0
30-R	2-9-79	750	900	7.5	8.0	753	350	618	61	113	21	345	427	.0
30-R	3-1-79	730	1000	7.6	8.2	668	350	463	37	90	18	309	427	.0
30-R	4-10-79	780	920	7.9	8.1	664	349	610	56	114	22	260	426	.0
30-R	5-14-79	770	760	8.3	7.9	682	369	527	46	100	29	279	450	.0
30-R	6-13-79	930	900	7.6	7.9	703	365	494	57	86	23	311	445	.0
30-R	7-18-79	1040	900	7.9	8.4	794	370	641	110	89	28	341	427	6.0
30-R	8-21-79	870	840	7.6	7.8	706	363	521	48	98	23	322	422	.0
30-R	11-14-79	1400	1170	7.4	7.7	770	379	628	53	120	30	332	462	.0
30-R	2-27-80	960	940	7.5	7.8	670	418	564	56	103	25	227	509	.0
30-R	5-27-80	900	830	7.5	7.6	840	650	652	58	123	28	234	793	.0
30-R	9-4-80	900	950	7.3	7.6	629	390	506	62	85	37	202	476	.0
30-R	12-10-80	790	880	7.1	7.4	709	434	545	78	85	35	245	529	.0
31-0	4-28-77	2300	2200	7.0	8.0	2064	620	1312	147	230	108	1202	756	.0
31-0	7-24-77	2160	2200	7.0	8.3	2149	665	1530	196	252	115	1166	811	.0
31-0	12-15-77	2390	2200	7.2	8.3	2126	630	1614	228	254	155	1104	769	.0
31-0	7-12-78	2260	2100	7.1	8.2	1917	664	1348	193	210	134	974	810	.0
31-0	8-23-78	1700	2200	7.2	8.0	2044	673	1494	189	248	134	1059	820	.0
31-0	9-29-78	2400	2150	7.1	8.2	1992	690	1520	191	253	118	1007	842	.0
31-0	11-29-78	1880	1700	7.0	7.8	2185	665	1482	169	257	245	1108	811	.0
31-0	2-28-79	1710	2400	7.1	7.7	1947	685	1128	132	194	129	1073	836	.0
31-0	4-10-79	2110	2000	7.5	8.0	1957	677	1426	196	227	120	986	825	.0
31-0	5-14-79	1990	1950	6.9	7.7	1880	704	1096	140	181	136	986	859	.0
31-0	6-13-79	2140	1950	7.0	7.5	1933	693	1083	156	168	127	1050	845	.0
31-0	7-18-79	2300	1620	7.1	8.2	2005	708	1134	180	166	127	1090	863	.0
31-0	8-21-79	1930	1800	7.0	7.6	1962	690	1102	144	180	116	1090	842	.0

Table 3C (continued)

Well Number	Date Sampled	Specific (field) Conductance µmhos/cm	Specific (lab) Conductance µmhos/cm	pH (field)	pH (lab)	Total Dissolved Solids (Calc.)	Alkalinity as $(CaCO_3)$	Hardness as $(CaCO_3)$	Calcium (Ca)	Magnesium (Mg)	Sodium (Na)	Sulfate (SO_4)	Bicarbonate (HCO_3)	Carbonate (CO_3)
						milligrams per liter								
31-0	11-14-79	2600	2300	7.0	7.5	1816	613	1205	134	212	136	949	747	.0
31-0	2-27-80	2200	2000	6.9	7.3	1708	820	1327	152	230	106	713	1000	.0
31-0	5-28-80	2300	1920	7.0	7.0	2000	713	1520	150	278	139	993	869	.0
31-0	9-4-80	2000	2100	7.1	7.1	1826	673	1115	95	213	147	950	821	.0
31-0	12-10-80	-	1950	6.8	7.1	1953	730	1335	179	215	121	986	889	.0
31-R	4-28-77	2110	2050	7.1	8.2	1874	485	1221	150	206	108	1114	592	.0
31-R	7-24-77	2070	2150	7.4	8.3	2129	563	1445	195	232	138	1204	686	.0
31-R	12-15-77	2480	2400	7.2	8.4	2203	575	1464	218	224	174	1364	653	12.0
31-R	7-12-78	2480	2400	7.2	8.3	2125	578	1353	220	195	167	1189	705	.0
31-R	8-23-78	2510	2250	7.3	7.9	2217	660	1447	200	230	164	1220	805	.0
31-R	9-29-78	2220	2100	6.8	8.1	2049	580	1414	190	228	133	1140	714	.0
31-R	11-29-78	1940	1800	7.2	7.9	2300	575	1384	170	233	283	1262	702	.0
31-R	2-28-79	1880	2500	7.0	7.9	2011	577	1088	142	178	156	1182	703	.0
31-R	4-10-79	2200	2100	7.3	7.5	2166	568	1475	231	218	159	1202	692	.0
31-R	5-14-79	2100	2050	7.1	8.2	2061	595	1105	162	170	172	1184	726	.0
31-R	6-13-79	2260	2260	7.0	7.8	2144	580	1149	183	168	164	1267	708	.0
31-R	7-18-79	2430	2000	7.2	8.0	2296	563	1578	368+	160	159	1253	686	.0
31-R	8-21-79	2100	1900	7.1	7.6	2064	582	1132	165	175	151	1208	709	.0
31-R	11-14-79	2700	2220	7.1	7.6	1988	578	1199	154	198	179	1097	705	.0
31-R	2-27-80	2300	1990	7.0	7.2	1866	615	1293	173	209	137	962	750	.0
31-R	5-28-80	2300	1970	7.0	7.3	1790	663	761	110	118	342	810	808	.0
31-R	9-4-80	2100	2300	7.1	7.2	1763	584	1091	105	202	164	927	712	.0
31-R	12-10-80	1600	1900	6.8	7.3	2112	600	1326	193	205	146	1195	732	.0
31-M	7-24-77	2140	2150	7.3	8.5	2186	517	1168	195	165	246	1288	473	39.0
31-M	12-14-77	1960	2200	7.3	8.4	2146	540	1057	180	147	288	1219	586	18.0
31-M	7-12-78	2440	2100	7.4	8.5	1997	570	1024	195	129	288	1051	622	18.0
31-M	8-23-78	2630	2400	7.2	8.2	2246	558	1236	204	176	264	1262	680	.0
31-M	9-29-78	2450	2200	7.2	8.1	2127	570	1237	198	180	249	1152	695	.0
31-M	2-28-79	1900	2500	7.4	8.1	1959	550	867	136	128	250	1110	671	.0
31-M	4-10-79	2050	2200	7.4	8.0	2219	558	1118	195	153	270	1252	680	.0
31-M	5-14-79	2270	1900	7.3	7.8	1985	593	908	151	129	276	1057	723	.0
31-M	6-13-79	2310	1750	7.2	7.8	2117	573	938	180	118	287	1174	698	.0
31-M	7-18-79	2500	1850	7.4	8.2	2161	553	979	195	120	233	1265	674	.0
31-M	8-21-79	2130	2000	7.3	7.6	2079	567	858	144	121	254	1204	691	.0
31-M	11-14-79	2800	2480	7.2	7.6	1917	563	950	142	144	278	998	686	.0
31-M	2-27-80	2300	1970	7.1	7.4	1979	600	1063	165	158	272	1008	732	.0
31-M	5-28-80	2100	2100	7.2	7.1	2082	673	1571	182	271	169	1041	821	.0
31-M	9-4-80	2100	1990	7.3	7.3	1834	580	759	146	96	315	915	707	.0
31-M	12-10-80		1950	6.8	7.1	1807	730	1335	179	215	121	986	889	.0

Table 30. (continued)

Well Number	Date Sampled	Specific (field) Conductance µmhos/cm	Specific (lab) Conductance µmhos/cm	pH (field)	pH (lab)	Total Dissolved Solids (Calc.)	Alkalinity as ($CaCO_3$)	Hardness as ($CaCO_3$)	Calcium (Ca)	Magnesium (Mg)	Sodium (Na)	Sulfate (SO_4)	Bicarbonate (HCO_3)	Carbonate (CO_3)
						milligrams per liter								
32-0	4-26-77	3570	2800	7.7	8.3	2936	285	1725	201	297	204	2056	348	.0
32-0	7-23-77	2610	3150	7.6	8.5	3098	385	1787	243	286	281	2051	409	15.0
32-0	12-14-77	2610	2700	7.6	8.5	2882	347	1594	217	255	279	1926	375	12.0
32-0	7-13-78	3090	2600	7.4	8.5	2833	354	1654	242	255	264	1863	383	12.0
32-0	8-23-78	2870	2650	7.4	8.3	2616	349	1716	242	270	240	1644	426	.0
32-0	9-29-78	2790	2600	7.4	8.1	2539	350	1605	219	257	225	1619	427	.0
32-0	11-30-78	2780	1900	7.0	8.3	2716	353	1527	178	263	462+	1591	430	.0
32-0	2-9-79	1930	2200	7.4	7.9	2302	355	1383	205	212	217	1444	433	.0
32-0	2-28-79	1950	2500	7.4	7.8	2206	350	1076	134	180	201	1474	427	.0
32-0	4-10-79	2280	2400	7.6	7.8	2444	346	1402	202	218	208	1590	422	.0
32-0	5-14-79	920+	2500	8.0	8.1	2612	354	1211	160	197	252	1774	432	.0
32-0	6-12-79	2600	2000	7.4	7.7	2818	360	1313	204	195	264	1922	439	.0
32-0	7-18-79	2800	2000	7.6	8.3	2606	355	1292	219	181	229	1745	433	.0
32-0	11-14-79	3200	2600	7.3	7.9	2153	354	1203	165	192	249	1320	431	.0
32-0	2-27-80	2200	1960	7.3	7.9	2044	404	1187	152	196	214	1225	493	.0
32-0	5-27-80	2200	1980	7.3	7.5	2188	425	1466	175	250	201	1291	519	.0
32-0	12-10-80	1900	1710	7.0	7.6	2034	375	1233	176	193	152	1274	458	.0
32-R	4-26-77	1440	1330	7.8	8.3	1050	308	805+	105	132	59	562	375	.0
32-R	7-22-77	1150	1120	7.4	8.3	875	448	633	105	90	78	322	546	.0
32-R	12-14-77	1030	1200	7.4	8.5	849	450	614	102	87	80	321	488	15.0
32-R	7-13-78	1200	1000	7.3	8.5	855	465	584	105	78	84	305	543	6.0
32-R	8-23-78	1170	1100	7.2	8.0	848	464	604	100	86	88	291	566	.0
32-R	9-29-78	1220	1100	7.0	8.2	830	450	645	100	96	79	280	549	.0
32-R	11-30-78	1180	880	7.3	8.3	943	460	562	88	83	160+	332	561	.0
32-R	2-9-79	910	1100	7.2	7.8	942	445	611	110	82	89	390	543	.0
32-R	2-28-79	870	1350	7.1	7.8	881	449	451	70	67	87	385	548	.0
32-R	4-10-79	1080	1220	7.4	7.6	925	448	615	101	88	85	370	546	.0
32-R	5-14-79	1190	1050	7.7	7.5	922	460	492	78	72	97	388	561	.0
32-R	6-12-79	1080	980	7.2	7.7	950	468	564	108	71	93	386	570	.0
32-R	7-18-79	1280	1000	7.4	8.1	956	458	548	112	65	85	406	558	.0
32-R	8-23-79	1115	1050	7.2	8.0	992	499	580	84	90	78	428	608	.0
32-R	11-14-79	1550	1290	7.1	8.0	879	459	539	78	84	96	336	559	.0
32-R	2-27-80	1200	1180	7.2	7.5	892	487	644	95	99	76	318	593	.0
32-R	5-28-80	1200	1220	7.3	7.1	1095	463	757	105	120	112	468	564	.0
32-R	9-4-80	900	1250	7.0	7.3	963	438	639	100	95	92	401	534	.0
32-R	12-10-80	1000	1150	7.3	7.6	985	475	633	108	88	83	411	580	.0

Table 3Q (continued)

Well Number	Date Sampled	Specific (field) Conductance µmhos/cm	Specific (lab) Conductance µmhos/cm	pH (field)	pH (lab)	Total Dissolved Solids (Calc.)	Alkalinity as $(CaCO_3)$	Hardness as $(CaCO_3)$	Calcium (Ca)	Magnesium (Mg)	Sodium (Na)	Sulfate (SO_4)	Bicarbonate (HCO_3)	Carbonate (CO_3)
						milligrams per liter								
33-0	4-29-77	965	870	10.7	9.4	636+	47+	374+	78	44+	53	431	21+	9.0
33-0	7-25-77	1150	1140	8.4	8.5	974	265	719	105	111	56	535	299	6.0
33-0	12-15-77	1110	1150	7.8	8.6	916	390	694	82	118	54	436	415	15.0
33-0	6-22-78	1360	1200	7.5	8.5	997	377	876	104	150	63	468	412	12.0
33-0	7-13-78	1210	1200	7.5	8.7	962	387	719	95	117	62	470	375	24.0
33-0	8-23-78	1280	1300	7.5	8.5	1035	382	802	98	135	68	512	400	16.5
33-0	9-28-78	1270	1200	7.3	8.2	1006	355	840	99	144	63	475	433	.0
33-0	11-30-78	1190	1000	7.6	8.4	1040	373	740	82	130	112	490	430	6.0
33-R	4-29-77	1620	1480	8.4	8.4	1245	280	724	102	114	102	748	342	.0
33-R	7-25-77	1830	1900	7.4	8.3	1674	468	1114	166	169	133	904	570	.0
33-R	12-15-77	1730	1900	7.6	8.4	1555	438	991	147	151	132	860	509	6.0
33-R	6-22-78	1940	1800	7.5	8.2	1654	454	1111	180	160	161	870	554	.0
33-R	7-11-78	1850	1600	7.3	8.5	1332	420	885	140	130	125	686	476	9.0
33-R	8-23-78	1660	1550	7.5	8.3	1443	513	947	140	145	130	711	625	.0
33-R	9-29-78	1620	1450	7.3	8.1	1245	380	861	122	135	101	648	464	.0
33-R	11-29-78	2000	1500	7.1	8.3	1730	445	1006	139	160	281+	874	543	.0
34-R	4-29-77	2110	2170	7.1	8.2	1936	543	1103	170	165	149	1113	662	.0
34-R	6-30-77	2240	2200	7.1	7.8	2346	600	1577	277	215	189	1280	732	.0
34-R	6-30-77	2460	2400	7.0	7.7	2287	605	1585	275	218	188	1220	738	.0
34-R	7-22-77	2800	2250	7.1	8.2	2377	593	1502	255	210	190	1337	723	.0
41-R	11-30-78	1320	1200	7.2	8.6	1235	435	793	106	128	188	558	458	18.0
41-R	3-1-79	1030	1600	7.5	7.7	1066	455	533	78	82	97	526	555	.0
41-R	4-10-79	1110	1350	7.5	7.7	1048	490	783	116	120	96	405	598	.0
41-R	5-14-79	1270	1250	7.2	7.7	1099	485	623	86	99	110	498	592	.0
41-R	6-12-79	1500	1350	7.3	7.8	1194	478	685	117	95	105	575	583	.0
41-R	7-18-79	1480	1110	7.5	8.1	1166	490	742	124	105	109	516	598	.0
41-R	8-21-79	1270	1250	7.4	8.0	1120	512	625	78	104	92	522	624	.0
41-R	11-14-79	1850	1460	7.2	8.1	1067	475	779	130	110	111	413	580	.0
41-R	5-28-80	1400	1440	7.3	7.5	1310	500	979	136	155	127	578	610	.0
41-R	9-4-80	-	1450	7.5	7.2	1132	478	665	74	117	115	523	583	.0
41-R	12-10-80	1300	1300	6.9	7.5	1161	504	727	102	115	95	533	614	.0
41-M	3-1-79	890	1500	11.7	11.1	996	388	150	55	3	144	413	22	365.0
41-M	5-14-79	880	920	10.3	10.2	696	480	37+	6	6	234	279	64	129.0
41-M	6-12-79	1040	940	9.7	9.4	740	384	72	16	8	240	302	189	69.0
41-M	7-18-79	1030	890	9.2	9.0	749	287	118	20	16	228	321	259	22.0
41-M	8-21-79	980	591	8.8	8.7	746	321	199	18	38	205	296	331	15.0

Table 30 (continued)

Well Number	Date Sampled	Specific (field) Conductance µmhos/cm	Specific (lab) Conductance µmhos/cm	pH (field)	pH (lab)	Total Dissolved Solids (Calc.)	Alkalinity as $(CaCO_3)$	Hardness as $(CaCO_3)$	Calcium (Ca)	Magnesium (Mg)	Sodium (Na)	Sulfate (SO_4)	Bicarbonate (HCO_3)	Carbonate (CO_3)
						milligrams per liter								
41-M	11-14-79	1300	1050	8.9	8.6	709	340	98	16	14	248	236	344	17.4
41-M	2-27-80	1000	820	8.8	8.8	720	400	65	12	9	222	248	397	22.5
41-M	5-28-80	1000	1000	8.4	8.5	709	280	150	27	20	245	256	268	18
41-M	9-4-80	-	1050	7.4	8.3	804	372	151	29	19	252	271	453	.0
41-0	3-1-79	1070	1550	7.7	8.2	1123	190	547	87	80	100	729	232	.0
41-0	4-10-79	1210	1400	8.5	8.1	1300	265	850	124	131	102	759	323	.0
41-0	5-14-79	1360	1300	7.9	8.2	1182	278	657	90	105	114	684	339	.0
41-0	6-12-79	1550	1400	8.0	8.1	1306	363	720	106	110	111	740	442	.0
41-0	7-18-79	1530	1220	8.0	8.1	1245	325	784	122	116	112	677	396	.0
41-0	8-21-79	1370	1320	7.6	8.2	1288	402	675	74	120	91	740	490	.0
41-0	11-14-79	1860	1610	7.3	7.7	1396	693	881	129	136	98	596	845	.0
41-0	2-27-80	1500	1750	7.5	7.4	1242	467	874	103	150	89	599	569	.0
41-0	5-28-80	1700	1930	7.6	7.4	1430	375	1112	150	179	134	724	458	.0
41-0	9-4-80	-	1550	7.2	7.4	1191	384	804	95	137	106	601	468	.0
41-0	12-10-80	1400	1450	7.1	7.2	1238	444	836	107	139	95	613	541	.0
42-0	4-11-79	770	870	7.6	7.9	607	353	488	80	70	34	202	430	.0
42-0	5-15-79	850	850	7.4	8.0	608	373	418	68	60	44	202	454	.0
42-0	6-12-79	770	800	7.4	7.6	615	368	430	76	58	35	217	448	.0
42-0	7-18-79	780	640	7.5	8.0	683	366	584	143	55	37	217	447	.0
42-0	8-21-79	780	700	7.5	8.2	558	370	329	51	49	30	197	451	.0
42-0	11-14-79	950	880	7.5	8.1	560	401	455	74	66	33	137	488	.0
42-0	2-27-80	790	750	7.4	7.6	569	388	475	66	75	27	161	473	.0
42-0	12-10-80	720	720	7.3	7.6	587	421	437	71	63	31	164	513	.0
42-R	4-11-79	830	1000	7.6	8.2	754	373	526	90	73	61	296	454	.0
42-R	5-15-79	920	800	7.6	8.2	655	467	394	69	54	72	171	569	.0
42-R	6-12-79	820	800	7.5	8.1	633	475	405	78	51	62	148	580	.0
42-R	7-18-79	840	600	7.4	8.1	633	460	468	90	59	64	134	561	.0
42-R	8-21-79	835	800	7.5	8.1	620	460	338	58	46	53	177	561	.0
42-R	11-14-79	1150	910	7.4	7.7	604	413	488	88	65	49	146	503	.0
42-R	2-27-80	870	720	7.4	7.3	587	425	478	68	75	38	144	519	.0
42-R	5-28-80	910	850	7.4	7.3	882	670	608	98	88	67	219	817	.0
42-R	9-4-80	700	880	7.2	7.4	641	397	469	68	73	54	200	483	.0
42-R	12-10-80	790	730	7.3	7.5	677	457	457	73	67	41	216	557	.0
42-M	4-11-79	870	1000	7.6	7.6	720	458	346	66	44	137	188	558	.0
42-M	5-15-79	1000	860	7.5	7.7	710	470	274	52	35	158	172	573	.0
42-M	6-12-79	920	750	7.9	8.3	740	448	290	62	33	159	208	546	.0
42-M	7-18-79	910	790	7.5	7.8	732	465	162	63	36	150	192	567	.0
42-M	8-21-79	940	900	7.6	8.2	691	463	224	45	27	141	190	564	.0

Table 30. (continued)

Well Number	Date Sampled	Specific (field) Conductance µmhos/cm	Specific (lab) Conductance µmhos/cm	pH (field)	pH (lab)	Total Dissolved Solids (Calc.)	Alkalinity as ($CaCO_3$)	Hardness as ($CaCO_3$)	Calcium (Ca)	Magnesium (Mg)	Sodium (Na)	Sulfate (SO_4)	Bicarbonate (HCO_3)	Carbonate (CO_3)
						milligrams per liter								
42-M	11-14-79	1200	1020	7.5	7.5	691	490	249	46	33	153	154	598	.0
42-M	2-27-80	930	900	7.5	7.5	657	497	307	57	39	130	121	605	.0
42-M	5-28-80	960	870	7.4	7.4	789	515	367	68	48	187	168	628	.0
42-M	9-4-80	700	970	7.3	7.5	697	475	273	51	36	150	165	580	.0
42-M	12-10-80	850	910	7.2	7.5	736	488	293	61	34	140	199	595	.0
43-0	3-1-79	1570	2300	7.0	8.0	1906	698	1061	141	172	125	1041	851	.0
43-0	4-10-79	1740	2000	7.1	8.0	2040	687	1265	170	204	116	1121	837	.0
43-0	5-15-79	1420	1450	7.2	7.9	1317	449	733	94	121	95	720	548	.0
43-0	6-12-79	2070	1900	7.1	7.8	1941	598	1096	168	164	137	1099	729	.0
43-0	7-18-79	2090	1610	7.2	7.8	1784	495	1151	168	178	115	1011	604	.0
43-0	11-14-79	2080	1820	7.1	7.5	1452	400	1056	154	163	97	780	488	.0
43-0	2-27-80	1900	1660	7.2	7.4	1490	410	1156	137	198	80	816	500	.0
43-0	5-28-80	1500	1840	7.4	7.9	1594	648	1235	168	198	102	724	790	.0
43-0	9-4-80	1400	1500	7.0	7.9	1242	390	833	92	147	78	676	475	.0
43-0	12-9-80	1200	1300	6.9	7.5	1136	350	786	106	127	63	619	427	.0
43-R	3-1-79	1410	1900	7.7	8.0	1389	543	817	96	140	98	725	662	.0
43-R	4-10-79	1830	2200	7.3	8.1	2326	600	1509	261	208	152	1328	732	.0
43-R	5-15-79	2030	2000	7.1	7.8	1929	674	1035	170	148	170	1022	822	.0
43-R	6-12-79	2070	1750	7.2	8.0	1982	640	1104	207	142	161	1073	781	.0
43-R	7-18-79	2130	1750	7.2	7.5	1800	693	1059	191	141	152	886	845	.0
43-R	11-14-79	2420	2100	7.2	7.5	1644	655	1116	192	155	142	745	799	.0
43-R	2-27-80	2300	1630	7.0	7.3	1892	441	1255	176	198	140	1100	538	.0
43-R	5-28-80	2000	2400	7.1	7.6	1930	713	1453	222	218	179	871	869	.0
43-R	9-4-80	1900	1900	7.1	7.2	1754	688	1062	130	179	159	860	839	.0
43-R	12-9-80	1800	1930	6.9	7.4	1757	713	1129	173	169	144	833	869	.0
43-M	3-1-79	1440	2100	7.8	7.8	1610	539	810	106	132	166	875	657	.0
43-M	4-10-79	1620	1800	7.4	7.9	1719	623	1051	170	152	192	813	760	.0
43-M	5-15-79	1840	1750	7.2	7.9	1794	736	914	148	132	191	867	897	.0
43-M	6-12-79	2090	1650	7.2	7.9	1924	720	1017	190	131	183	972	878	.0
43-M	7-18-79	2150	1550	7.4	7.7	1832	680	1011	170	142	188	906	830	.0
43-M	11-14-79	2720	2070	7.1	7.5	1718	724	1107	194	151	183	738	882	.0
43-M	2-27-80	1900	1560	7.0	7.3	1394	751	876	130	134	129	729	519	.0
43-M	5-28-80	2000	2080	7.1	8.0	2024	751	1415	225	207	215	913	915	.0
43-M	9-4-80	1900	2100	7.1	7.6	1819	741	1025	129	171	189	872	903	.0
43-M	12-9-80	1900	1880	7.0	7.4	1768	749	1064	170	155	163	821	914	.0

Table 30. (continued)

Well Number	Date Sampled	Specific (field) Conductance µmhos/cm	Specific (lab) Conductance µmhos/cm	pH (field)	pH (lab)	Total Dissolved Solids (Calc.)	Alkalinity as ($CaCO_3$)	Hardness as ($CaCO_3$)	Calcium (Ca)	Magnesium (Mg)	Sodium (Na)	Sulfate (SO_4)	Bicarbonate (HCO_3)	Carbonate (CO_3)
						milligrams per liter								
CM-1	2-9-79	1280	1450	6.9	8.1	1267	285	716	132	94	152	709	348	.0
CM-1	2-28-79	1150	1700	7.4	8.0	1191	369	582	101	80	133	648	450	.0
CM-1	4-10-79	100+	1320	7.5	8.2	1155	378	738	130	100	120	562	461	.0
CM-1	5-13-79	1430	1500	7.4	7.6	1206	385	724	136	93	140	589	470	.0
CM-1	6-13-79	1580	1350	7.4	7.8	1293	403	730	156	82	134	665	491	.0
CM-1	7-18-79	1450	1200	7.4	8.3	1167	378	662	143	74	129	577	461	.0
CM-1	8-21-79	1365	1400	7.5	8.2	1156	372	552	98	74	104	640	453	.0
CM-1	11-14-79	1750	1920	7.4	8.1	1170	420	713	120	100	130	547	512	.0
CM-1	2-27-80	1400	1320	7.5	7.7	1008	491	665	115	92	88	399	599	.0
CM-1	5-27-80	1100	1200	7.2	7.6	1086	463	726	110	109	131	444	564	.0
CM-1	9-4-80	1100	1180	7.4	7.1	927	400	583	103	79	113	379	487	.0
CM-1	12-9-80	990	1080	7.2	7.1	972	425	540	101	70	109	426	519	.0
CR-2	11-30-78	4160	3000	7.5	8.0	4548	255	1970	96+	420	727+	3100	311	.0
CR-2	2-9-79	3410	3600	7.1	8.1	4450	248	2470	354	385	362	3146	302	.0
CR-2	2-28-79	3210	3900	7.4	8.0	4411	293	2147	292	344	356	3192	357	.0
CR-2	4-10-79	100+	3500	7.7	8.1	4759	288	2570	386	390	394	3350	351	.0
CR-2	5-13-79	3570	3200	7.4	7.9	3977	308	2146	324	325	336	2749	375	.0
CR-2	6-13-79	4100	3000	7.5	7.5	4471	295	2269	384	319	384	3148	360	.0
CR-2	7-18-79	3840	3500	7.6	8.1	4315	291	2044	365	275	351	3089	355	.0
CR-2	2-28-80	3500	3300	7.6	7.3	3196	335	1910	270	300	319	2050	409	.0
CR-2	5-27-80	3900	3790	7.3	7.6	4513	360	2583	325	430	360	3119	439	.0
CR-2	9-4-80	4400	3490	7.4	7.4	4805	325	2958	426	460	412	3248	397	.0
CR-2	12-9-80	3300	3960	6.8	7.5	4933	328	2568	355	408	357	3553	400	.0
D-2	7-14-78	1870	2100	7.3	8.3	1850	453	1207	173	188	164	1036	552	.0
D-2	8-22-78	1880	2100	7.2	8.1	1967	432	1268	181	198	152	1162	526	.0
D-2	9-28-78	2010	2000	7.4	8.3	2025	453	1407	202	219	151	1164	522	.0
ODS*-1	4-17-77	765	900	7.4	7.9	624	325	521	60	87	29	245	396	.0
OBD**	4-20-77	520	660	8.0	8.2	378	288	341	38	60	16	91	351	.0

*Dip sample from overburden blast hole approximately 90 m NE of Well 29-R

**Spring discharge from highwall 30 cm above top of coal seam

+Chemical value not considered representative and therefore not utilized in determining range and mean

Table 31. Chemical analyses showing minor and trace element constituents in surface and ground water collected from Area B of the Western Energy Company mine near Colstrip, Montana.

Well Number	Date Sampled	Iron (Fe)	Copper (Cu)	Zinc (Zn)	Manganese (Mn)	Lead (Pb)	Cadmium (Cd)	Boron (B)	Nickel (Ni)	Fluoride (F)	Chloride (Cl)	Nitrate (NO_3-N)	Phosphorous (Total P)	Potassium (K)	SAR
		milligrams per liter													
E.F. Armells	5-19-77	<.010	.495	.036	.057	<.010	.015	.28	7.30	.30	<.01	.26	<.01	-	1.10
E.F. Armells	3-10-78	.050	.045	<.005	.012	<.010	<.005	<.01	1.40	<.01	.01	.04	<.01	-	.20
E.F. Armells	3-11-78	.056	.049	.010	.010	<.010	<.005	<.01	1.40	.02	.01	.03	<.01	-	.20
E.F. Armells	3-13-78	.238	.178	.008	.017	<.010	<.005	<.01	2.80	.14	.10	.03	<.01	-	.23
E.F. Armells	3-17-78	.270	.452	.010	.027	<.010	<.005	.07	3.80	<.01	.07	.07	<.01	-	.96
E.F. Armells	4-18-78	.080	.009	.023	.591	<.010	<.005	<.01	<.01	.20	17.40	<.01	.02	-	1.98
E.F. Armells	5-08-78	.056	.013	.018	.170	<.010	<.005	.32	<.01	.30	6.60	.35	.01	-	1.00
E.F. Armells	5-19-78	.062	.010	.020	.018	<.010	<.005	.40	<.01	.27	7.50	<.01	<.01	-	1.10
E.F. Armells	5-19-78	.070	.021	.005	.148	<.010	<.005	.45	<.01	.24	9.90	.03	.01	-	1.00
E.F. Armells	7-13-78	.042	.022	.022	.334	<.010	<.005	.85	<.01	.20	15.50	<.01	.02	-	2.16
E.F. Armells	8-22-78	.030	.020	.014	.120	.012	.008	.63	<.01	.20	20.20	.03	.04	-	2.40
E.F. Armells	4-10-79	.012	<.010	.030	.341	.030	<.005	.44	<.01	.28	11.80	.06	.01	10.1	1.80
E.F. Armells	5-14-79	.030	.060	.024	1.028	<.010	<.005	.42	<.01	.28	12.70	.29	.01	10.6	1.84
E.F. Armells	6-12-79	.048	.036	.030	1.345	<.010	<.005	.28	<.01	.27	13.60	.05	.01	10.6	2.03
E.F. Armells	7-16-79	.090	.018	.018	5.700	<.010	<.005	.60	<.01	.29	21.60	.05	.05	14.7	2.05
E.F. Armells	8-23-79	<.010	.028	<.005	.146	<.010	<.005	.75	<.01	.28	29.60	.03	.04	19.0	3.07

(continued on following page)

Table 31. (continued)

Well Number	Date Sampled	Iron (Fe)	Copper (Cu)	Zinc (Zn)	Manganese (Mn)	Lead (Pb)	Cadmium (Cd)	Boron (B)	Nickel (Ni)	Fluoride (F)	Chloride (Cl)	Nitrate (NO_3-N)	Phosphorous (Total P)	Potassium (K)	SAR
		milligrams per liter													
Impoundment	7-13-78	<.020	.018	.022	<.010	<.010	<.005	.45	<.010	.20	6.6	.24	.01	-	1.63
Impoundment	8-23-78	<.010	.010	.014	<.010	<.010	<.005	.47	<.010	.20	5.2	.53	.02	-	1.40
Impoundment	9-29-78	.040	.011	.010	.005	<.010	.007	.53	.012	.20	5.6	.29	.01	-	1.20
Impoundment	3-2-79	.032	.020	.015	<.010	<.010	<.005	.08	<.010	.12	1.4	<.01	.02	-	0.37
Impoundment	4-10-79	<.010	<.010	.014	.008	<.010	<.005	.11	<.010	.14	0.9	.01	.04	2.0	0.60
Impoundment	5-14-79	.015	.042	.010	.012	<.010	<.005	.38	<.010	.19	5.2	.15	<.01	5.8	1.51
Impoundment	6-12-79	.020	.030	.010	.010	<.010	<.005	.36	<.010	.21	7.5	<.01	.01	7.4	1.59
Impoundment	7-17-79	.050	.015	.014	.036	<.010	<.005	.43	<.010	.21	12.7	.01	.01	10.3	2.26
4-R	4-20-77	.020	.031	.225	.270	.015	<.005	.40	.090	.20	9.4	.09	.00	-	1.20
4-R	7-25-77	.054	.027	.292	.455	<.010	<.005	.28	.040	.26	7.1	.04	<.01	-	1.20
4-R	12-15-77	.022	.032	.260	.180	<.010	<.005	.51	.070	.22	5.2	<.01	.01	-	2.00
4-R	5-16-78	.070	.032	.598	.542	.012	<.005	.46	.010	.26	8.0	.02	.07	-	1.20
4-R	6-22-78	.056	.011	.298	.225	<.010	<.005	.30	<.010	.27	7.5	.06	.02	-	1.42
4-R	7-12-78	.046	.020	.180	.220	<.010	<.005	.43	<.010	.20	8.5	<.01	.09	-	1.48
4-R	8-17-78	.016	.015	.197	.196	<.010	<.005	.36	<.010	.30	8.9	.05	.02	-	1.50
4-M	4-20-77	.050	.007	.028	.095	.020	.010	.43	.033	.22	4.7	.09	.00	-	2.40
4-M	7-25-77	.055	.023	.019	.208	<.010	<.005	.38	<.010	.24	4.7	<.01	.01	-	2.50
4-M	12-15-77	.020	.020	.090	.130	<.010	<.005	.56	.010	.18	4.2	.09	<.01	-	1.80
4-M	5-16-78	.054	.015	.330	.180	<.010	<.005	.47	<.010	.20	5.2	<.01	<.01	-	2.80
4-M	6-22-78	.076	.010	.032	.108	<.010	<.005	.44	<.010	.23	6.1	.06	.01	-	2.88
4-M	7-12-78	.070	.020	.069	.120	<.010	<.005	.50	<.010	.20	6.1	.08	.01	-	3.02
4-M	8-17-78	.086	.022	.040	.098	<.010	<.005	.52	<.010	.20	3.3	.07	.01	-	2.90
5-R	4-28-77	.110	.019	.029	.176	.012	<.005	.40	.039	.18	18.3	.14	.00	-	1.70
5-R	7-22-77	.074	.030	.438	.547	<.010	<.005	.24	.018	.17	16.9	.03	.01	-	1.60
7-0	4-27-77	.060	.019	.039	.206	.015	<.005	.32	.013	.20	8.0	.01	.02	-	1.10
7-R	4-27-77	.030	.012	.026	.156	.015	.012	.32	.021	.20	6.6	.03	.00	-	1.80
7-M	4-27-77	.08	.012	.060	.102	<.010	.008	.37	.020	.24	5.2	.09	.00	-	2.70
10-R	7-24-77	.069	.037	.038	.608	<.010	<.005	.49	<.010	.17	7.5	.05	<.01	-	2.80
10-R	7-13-78	.072	.020	.270	.242	<.010	<.005	.65	<.010	.10	6.1	.10	.01	-	3.51
10-R	8-22-78	.054	.024	.200	.260	<.010	<.005	.62	<.010	.20	4.7	.01	.02	-	3.60
10-R	9-28-78	.058	.009	.391	.134	<.010	<.005	.62	.010	.10	5.2	.18	.01	-	3.27
10-R	3-1-79	.044	.022	.551	.200	<.010	<.005	.58	<.010	.12	4.2	<.01	.01	-	3.64
10-R	4-10-79	.018	.020	.120	.230	<.010	<.005	.64	<.010	.15	5.2	.02	.01	10.2	3.60
10-R	5-14-79	.028	.030	.109	.221	<.010	<.005	.52	<.010	.13	3.3	.12	<.01	10.0	3.97

(continued on following page)

Table 31. (continued)

Well Number	Date Sampled	Iron (Fe)	Copper (Cu)	Zinc (Zn)	Manganese (Mn)	Lead (Pb)	Cadmium (Cd)	Boron (B)	Nickel (Ni)	Fluroide (F)	Chloride (Cl)	Nitrate (NO_3-N)	Phosphorous (Total P)	Potassium (K)	SAR
		milligrams per liter													
10-R	6-13-79	.040	.026	.130	.270	<.010	<.005	.42	<.010	.12	3.3	.12	.01	9.1	3.96
10-R	7-18-79	.060	.020	.321	.347	<.010	<.005	.53	<.010	.12	8.0	.33	.01	10.7	3.91
10-R	8-21-79	.010	.010	.235	.309	<.010	<.005	.51	<.010	.11	6.6	.08	.04	10.4	3.94
10-R	11-14-79	<.010	.026	.200	.378			.62		.14	7.5	.16	.08	8.72	4.39
10-R	2-27-80	.048	.047	.162	.294			.37		.12	10.8	.29	.03	11.7	3.34
10-R	5-28-80	.050	.047	.204	.417			.45		.11	9.3	.25	.05	10.30	2.93
10-R	9-4-80	.092	.058	.120	.316			.39		.10	11.3	.33	.02	10.42	3.26
10-R	12-10-80	.121	.042	.119	.273			.41		.06	8.5	.18	.01	9.22	3.01
15-0	4-29-77	.082	.018	.007	.009	<.010	.014	.17	.018	.24	4.20	<.01	.10	-	2.20
15-0	7-23-77	.069	.038	.058	.009	<.010	<.005	.05	<.010	.26	10.80	.05	<.01	-	2.20
15-0	7-12-78	.322	.023	.148	.520	<.010	<.005	.43	<.010	.10	8.0	.04	.02	-	2.14
15-0	8-22-78	.016	.036	.333	.280	<.010	<.005	.49	<.010	.10	8.0	.32	.03	-	2.10
15-0	8-29-78	.036	.011	.158	.257	<.010	<.005	.87	.014	.10	8.5	.07	.02	-	1.93
15-0	11-30-78	.040	.023	.446	.268	<.010	<.005	.39	<.010	.03	8.9	.03	<.01	-	4.01
15-0	2-28-79	.026	.032	.592	.213	<.010	<.005	.51	<.010	.12	8.5	<.01	.02	-	2.20
15-0	4-10-79	.020	.024	.161	.193	<.010	<.005	.60	<.010	.14	7.5	.05	.01	12.2	2.00
15-0	5-14-79	.034	.025	.155	.149	<.010	<.005	.33	<.010	.14	7.1	.13	.01	12.4	2.45
15-0	6-12-79	.040	.060	.238	.170	<.010	<.005	.35	<.010	.13	8.9	.20	.01	11.0	2.35
15-0	7-18-79	.065	.020	.192	.183	<.010	<.005	.45	<.010	.12	11.3	.10	.02	12.1	2.26
15-0	8-23-79	<.010	.010	.230	.184	<.010	<.005	.50	.013	.12	10.3	.04	.05	12.5	2.08
15-0	11-14-79	<.010	.018	.219	.177			.53		.14	9.6	<.01	.07	8.36	2.36
15-0	2-27-80	.053	.060	.159	.163			.28		.13	8.8	.47	.01	12.2	2.16
15-0	5-28-80	.045	.060	.240	.230			.36		.12	7.4	.18	-	10.20	2.14
15-0	9-4-80	.072	.060	.188	.154			.34		.10	8.3	.24	.11	10.82	2.21
15-0	12-10-80	.058	.044	.151	.146			.36		.06	5.6	.31	.01	9.90	1.90
15-R	7-23-77	.069	.028	.033	.321	<.010	<.005	.39	.018	.17	5.6	.03	<.01	-	2.50
15-R	7-12-78	.100	.020	.240	.142	<.010	<.005	.50	<.010	.10	10.8	<.01	.01	-	1.76
15-R	8-22-78	.012	.016	.200	.174	<.010	<.005	.55	<.010	.10	13.2	.03	.02	-	1.70
15-R	9-29-78	.028	.010	.095	.160	<.010	<.005	.49	.012	.10	10.4	.06	.02	-	1.47
15-R	11-30-78	.040	.013	.150	.147	<.010	<.005	.31	<.010	.01	11.8	.03	<.01	-	2.72
15-R	2-28-79	.020	.022	.191	.126	<.010	<.005	.52	<.010	.08	10.3	<.01	.01	-	1.43
15-R	4-10-79	.019	.022	.143	.157	<.010	<.005	.48	<.010	.08	11.8	<.01	<.01	8.0	1.30
15-R	5-14-79	.019	.028	.124	.132	<.010	<.005	.26	<.010	.08	11.8	.06	<.01	8.0	1.65
15-R	6-12-79	.040	.030	.110	.165	<.010	<.005	.39	<.010	.08	11.8	.03	.01	7.7	1.53
15-R	7-18-79	.058	<.008	.090	.183	<.010	<.005	.36	<.010	.08	11.8	.02	.01	8.2	1.56
15-R	8-23-79	<.010	<.010	.072	.149	<.010	<.005	.40	<.010	.07	12.2	.07	.04	7.9	1.35
15-R	11-14-79	.015	.026	.076	.095			.54		.08	11.5	.01	.05	5.73	1.58
15-R	2-27-80	.064	.080	.076	.114			.28		.07	11.4	.07	.01	7.80	1.12

Table 31. (continued)

Well Number	Date Sampled	Iron (Fe)	Copper (Cu)	Zinc (Zn)	Manganese (Mn)	Lead (Pb)	Cadmium (Cd)	Boron (B)	Nickel (Ni)	Fluroide (F)	Chloride (Cl)	Nitrate (NO_3-N)	Phosphorous (Total P)	Potassium (K)	SAR
		milligrams per liter													
15-R	5-28-80	.030	.045	.108	.225			.36		.07	10.4	.06	.01	6.80	1.36
15-R	9-4-80	.068	.035	.080	.128			.29		.08	2.9	.30	.06	7.52	1.50
15-R	12-10-80	.053	.066	.062	.079			.36		.04	9.4	.27	.01	6.76	1.39
15-M	7-23-77	.092	.028	.113	.091	<.010	<.005	.39	.012	.19	11.3	.08	.01	-	1.60
15-M	7-11-78	.080	.018	.240	.175	<.010	<.005	.53	<.010	.10	9.9	.01	.01	-	3.31
15-M	8-22-78	.032	.030	.102	.136	<.010	<.005	.54	-	.10	7.1	.06	.02	-	3.20
15-M	9-29-78	.036	.010	.238	.170	<.010	<.005	.52	.014	.10	6.1	.06	.01	-	2.99
15-M	11-30-78	.036	.016	.443	.132	<.010	<.005	.40	<.010	.02	5.6	.04	<.01	-	6.21
15-M	2-28-79	.050	.036	.158	.144	<.010	<.005	.62	<.010	.14	6.6	.01	.01	-	3.21
15-M	4-10-79	.024	.025	.180	.147	<.010	<.005	.63	<.010	.10	7.5	<.01	<.01	11.5	3.10
15-M	5-14-79	.026	.020	.068	.127	<.010	<.005	.43	<.010	.09	4.2	.03	<.01	11.4	3.63
15-M	6-12-79	.045	.034	.089	.150	<.010	<.005	.47	<.010	.10	6.6	.08	.02	10.6	3.57
15-M	7-18-79	.060	.012	.138	.150	<.010	<.005	.48	<.010	.09	7.5	.01	<.01	11.7	3.46
15-M	8-23-79	<.010	.010	.249	.146	<.010	<.005	.57	<.010	.08	7.1	.01	.05	11.6	3.40
15-M	11-14-79	<.010	.033	.152	.082			.58		.10	7.1	<.01	.07	7.75	4.28
15-M	2-28-80	.051	.044	.155	.104			.32		.09	5.7	.21	.04	12.10	3.49
15-M	5-28-80	.048	.056	.104	.180			.38		.08	5.9	.21	.04	10.14	3.15
15-M	9-4-80	.052	.035	.110	.110			.35		.08	5.2	.19	.06	10.80	3.42
15-M	12-10-80	.058	.068	.153	.101			.35		.05	4.2	.25	<.01	9.64	3.26
28-0	4-29-77	.058	.016	.025	.038	<.010	.009	.23	.020	.22	6.6	.12	.13	-	1.00
28-R	4-29-77	.020	.010	.005	.028	<.010	.011	.17	.010	.24	5.2	.04	<.01	-	1.50
29-0	4-27-77	.022	.019	.005	.040	<.010	.011	.15	<.005	.22	.9	.01	.51	-	0.30
29-R	4-27-77	.022	.010	.017	.148	.015	<.005	.36	.005	.20	1.4	.03	<.01	-	0.80
30-0	4-29-77	.078	.010	.048	.117	<.010	<.005	.22	.012	.32	.9	.08	<.01	-	0.50
30-0	7-24-77	.074	.033	.040	.020	<.010	<.005	.12	<.010	.35	1.4	.40	.01	-	0.30
30-0	6-22-78	<.010	.008	.110	.050	<.010	<.005	.08	<.010	.33	0.9	.37	.02	-	0.37
30-0	7-13-78	.024	.020	.048	<.010	<.010	<.005	.08	<.010	.20	1.9	.02	.01	-	0.43
30-0	8-17-78	.018	.020	.028	.076	<.010	<.005	.14	<.010	.30	1.4	.19	.01	-	0.50
30-0	9-29-78	.018	<.009	.019	.061	.013	<.005	.21	<.010	.40	0.9	.05	.02	-	0.34
30-0	11-30-78	.025	.020	.043	<.010	<.010	<.005	.12	<.010	.16	1.4	.02	<.01	-	0.74
30-0	2-9-79	.030	.036	.070	.010	<.010	<.005	.25	<.010	.39	0.9	.33	.01	-	0.38
30-0	3-1-79	.020	.104	.111	.029	<.010	<.005	.20	<.010	.31	2.4	<.01	.01	-	0.37
30-0	4-10-79	<.010	.098	.060	.009	<.010	<.005	.26	<.010	.33	1.4	.06	.02	4.40	0.40
30-0	5-14-79	<.010	.100	.042	.019	<.010	<.005	.05	<.010	.36	1.4	.30	<.01	4.44	0.54
30-0	6-13-79	.035	.094	.051	<.005	<.010	<.005	.07	<.010	.38	1.4	.03	.01	4.08	0.39
30-0	7-18-79	.043	.036	.030	.043	<.010	<.005	.11	<.010	.36	2.4	-	<.01	4.90	0.43

Table 31. (continued)

Well Number	Date Sampled	Iron (Fe)	Copper (Cu)	Zinc (Zn)	Manganese (Mn)	Lead (Pb)	Cadmium (Cd)	Boron (B)	Nickel (Ni)	Fluroide (F)	Chloride (Cl)	Nitrate (NO_3-N)	Phosphorous (Total P)	Potassium (K)	SAR
		milligrams per liter													
30-R	4-29-77	.076	.010	.016	.050	.015	<.005	.25	.017	.22	.5	.06	.00	-	0.60
30-R	7-24-77	.065	.029	.042	.075	<.010	<.005	.19	<.010	.33	2.8	.16	.05	-	0.40
30-R	12-15-77	.010	.020	.145	.074	<.010	<.005	.32	<.010	.23	2.4	.27	.01	-	0.70
30-R	6-23-78	<.010	.008	.059	.095	<.010	<.005	.09	<.010	.25	0.9	.26	.02	-	0.40
30-R	7-13-78	<.020	.019	.042	<.010	<.010	<.005	.12	-	.30	2.4	.01	.02	-	0.49
30-R	8-22-78	<.010	.020	.056	.082	<.010	<.005	.19	<.010	.30	1.9	.53	.03	-	0.50
30-R	9-29-78	<.010	<.009	.067	.083	.010	.005	.16	.011	.30	1.4	.11	.02	-	0.34
30-R	11-29-78	.015	.015	.112	.083	<.010	<.005	.06	<.010	.07	2.4	.04	<.01	-	0.68
30-R	2-9-79	.056	.040	.075	.028	<.010	<.005	.33	<.010	.25	1.9	.13	.02	-	0.37
30-R	3-1-79	.024	.022	.149	.078	<.010	<.005	.28	<.010	.22	2.4	.11	.01	-	0.37
30-R	4-10-79	<.010	.019	.065	.052	<.010	<.005	.30	<.010	.25	1.9	.11	.01	4.60	0.40
30-R	5-14-79	<.010	.040	.063	.041	<.010	<.005	<.01	<.010	.24	1.9	<.01	<.01	4.78	0.55
30-R	6-13-79	.032	.030	.042	.072	<.010	<.005	.06	<.010	.23	2.4	.12	<.01	4.68	0.46
30-R	7-18-79	.045	.015	.050	.101	<.010	<.005	.12	<.010	.23	3.3	.26	.01	5.6	.48
30-R	8-17-79	.010	<.010	.050	.040	<.010	<.005	.17	<.010	.22	2.4	.03	.05	5.2	.43
30-R	11-14-79	<.010	.010	.040	.139			.15		.28	2.6	.12	.07	6.18	0.52
30-R	2-77-80	.065	.060	.060	.170			.10		.21	2.6	.01	.04	5.30	0.45
30-R	5-27-80	.035	.038	.054	.122			.18		.21	2.0	.17	.02	4.82	0.47
30-R	9-4-80	.064	.063	.448	.040			.13		.18	2.1	.37	.08	5.06	0.72
30-R	12-10-80	.042	.052	.043	.044			.26		.09	.5	.24	.03	4.68	0.65
31-0	4-28-77	.318	.015	<.005	.061	<.010	.012	.34	.020	.18	3.8	.03	<.01	-	1.30
31-0	7-24-77	.909	.038	.057	.113	<.010	<.005	.33	<.010	.22	6.6	.09	.01	-	1.30
31-0	12-15-77	.433	.028	.275	.088	.025	<.005	.54	.015	.18	5.2	.08	.01	-	1.68
31-0	7-12-78	.680	.022	.129	.123	<.010	<.005	.44	<.010	.20	6.1	<.01	.01	-	1.59
31-0	8-23-78	.630	.030	.091	.080	<.010	<.005	.40	<.010	.20	8.7	.02	.02	-	1.50
31-0	9-29-78	.570	<.009	.059	.120	<.010	<.005	.43	<.010	.20	7.5	.03	.01	-	1.32
31-0	11-29-78	.760	.020	.080	.175	<.010	<.005	.22	<.010	.03	6.6	<.01	<.01	-	2.77
31-0	2-28-79	.480	.030	.109	.136	<.010	<.005	.44	<.010	.15	6.1	<.01	.01	-	1.67
31-0	4-10-79	.684	.020	.082	.090	<.010	<.005	.38	<.010	.17	8.0	<.01	<.01	12.1	1.40
31-0	5-14-79	.832	.025	.044	.075	<.010	<.005	.16	<.010	.17	4.2	.02	<.01	8.6	1.79
31-0	6-13-79	1.100	.020	.033	.059	<.010	<.005	.32	<.010	.16	7.1	.02	.01	7.4	1.68
31-0	7-18-79	.300	.018	.063	.113	<.010	<.005	.39	<.010	.16	7.5	<.01	.01	8.9	1.64
31-0	8-21-79	.743	<.010	.056	.096	<.010	<.005	.45	<.010	.16	8.0	.09	.03	8.4	1.53
31-0	11-14-79	.410	.018	.033	.040			.41		.20	7.0	<.01	.05	9.50	1.71
31-0	2-27-80	.560	.022	.047	.074			.18		.15	6.4	.08	.05	8.50	1.26
31-0	5-28-80	.750	.040	.078	.168			.26		.15	4.0	.03	.02	7.80	1.55
31-0	9-4-80	.556	.045	.088	.076			.17		.14	7.5	.26	.04	8.20	1.92
31-0	12-10-80	1.184	.045	.037	.102			.29		.08	4.7	.08	.02	7.44	1.44

Table 31. (continued)

Well Number	Date Sampled	Iron (Fe)	Copper (Cu)	Zinc (Zn)	Manganese (Mn)	Lead (Pb)	Cadmium (Cd)	Boron (B)	Nickel (Ni)	Fluroide (F)	Chloride (Cl)	Nitrate (NO_3-N)	Phosphorous (Total P)	Potassium (K)	SAR
		milligrams per liter													
31-R	4-28-77	.120	.013	.064	.164	.020	.014	.43	.028	.20	4.7	.02	.00	–	1.30
31-R	7-24-77	.083	.036	.033	.255	<.010	<.005	.41	<.010	.24	5.6	.02	<.01	–	1.60
31-R	12-15-77	.018	.024	.188	.331	.010	<.005	.70	.015	.19	5.2	.50	<.01	–	1.99
31-R	7-12-78	.030	.020	.140	.330	<.010	<.005	.63	<.010	.20	5.2	<.01	.01	–	1.98
31-R	8-23-78	.028	.024	.094	.248	<.010	<.005	.58	<.010	.20	6.6	.06	.01	–	1.90
31-R	9-28-78	.020	.010	.062	.243	<.010	<.005	.46	<.010	.20	6.1	.17	.02	–	1.54
31-R	11-29-78	.060	.017	.101	.303	<.010	<.005	.40	<.010	.04	5.6	.18	.03	–	3.31
31-R	2-28-79	.024	.025	.134	.301	<.010	<.005	.58	<.010	.18	5.2	<.01	.02	–	2.06
31-R	4-10-79	.036	.024	.095	.260	<.010	<.005	.58	<.010	.20	5.6	.12	<.01	9.1	1.80
31-R	5-14-79	.032	.030	.056	.213	<.010	<.005	.36	<.010	.21	5.6	<.01	.01	9.4	2.25
31-R	6-13-79	.055	.024	.048	.248	<.010	<.005	.41	<.010	.18	4.7	.10	.01	8.4	2.10
31-R	7-18-79	.063	.018	.069	.250	<.010	<.005	.68	<.010	.18	7.5	.10	.03	9.6	1.74
31-R	8-21-79	.012	.015	.058	.274	<.010	<.005	.64	<.010	.19	6.1	.11	.04	9.2	1.95
31-R	11-14-79	.015	.014	.040	.157			.61		.24	5.4	.09	.07	6.80	2.26
31-R	2-27-80	.072	.078	.067	.163			.32		.19	5.8	.01	.02	9.40	1.66
31-R	5-28-80	.020	.030	.130	.132			.30		.34	3.6	.08	.01	8.40	5.40
31-R	9-4-80	.064	.080	.056	.238			.32		.18	6.3	.11	.07	8.68	2.15
31-R	12-10-80	.074	.048	.028	.237			.38		.09	5.2	.18	.04	7.50	1.74
31-M	7-24-77	.377	.028	.429	.076	<.010	<.005	.34	<.010	.31	4.7	.03	<.01	–	3.10
31-M	12-14-77	.775	.020	.390	.105	<.010	<.005	.58	.010	.32	4.7	.07	<.01	–	3.86
31-M	7-12-78	.060	.020	3.800+	.102	<.010	<.005	.50	<.010	.30	5.2	<.01	.02	–	3.93
31-M	8-23-78	.824	.026	.235	.078	<.010	<.005	.60	<.010	.30	4.7	.02	.02	–	3.30
31-M	9-28-78	.516	.012	.248	.077	<.010	<.005	.60	.013	.30	3.8	.09	.01	–	3.09
31-M	2-28-79	.060	.037	.453	.098	<.010	<.005	.72	<.010	.32	4.2	.12	.02	–	3.69
31-M	4-10-79	.020	.033	.545	.080	<.010	<.015	.62	<.010	.33	3.8	.06	<.01	9.8	3.50
31-M	5-14-79	.073	.032	.164	.066	<.010	<.005	.38	<.010	.34	4.7	.23	<.01	10.0	4.00
31-M	6-13-79	.115	.040	.170	.062	<.010	<.005	.35	<.010	.32	3.8	.26	<.01	9.1	4.08
31-M	7-18-79	.062	.016	.214	.099	<.010	<.005	.48	<.010	.31	4.7	.02	<.01	10.4	3.24
31-M	8-21-79	.015	.020	.308	.076	<.010	<.005	.51	<.010	.31	5.2	.02	.03	10.8	3.77
31-M	11-14-79	.020	.014	.228	.110			.56		.37	5.3	<.01	.08	10.10	3.93
31-M	2-27-80	.068	.041	.062	.171			.29		.31	5.0	.10	.04	10.00	3.63
31-M	5-28-80	.040	.041	.040	.310			.40		.17	6.2	.05	.02	8.34	1.89
31-M	9-4-80	.032	.035	.120	.042			.31		.34	6.0	.15	.03	8.28	4.98
31-M	12-10-80	.047	.061	.098	.050			.30		.18	4.2	.17	.04	7.62	5.09
32-0	4-26-77	.116	.018	.129	.129	<.010	.010	.20	.030	.26	.61	.42	.00	–	2.10
32-0	7-23-77	.083	.033	.238	.113	<.010	<.005	.25	<.010	.26	9.4	.29	.02	–	2.90

Table 31. (continued)

Well Number	Date Sampled	Iron (Fe)	Copper (Cu)	Zinc (Zn)	Manganese (Mn)	Lead (Pb)	Cadmium (Cd)	Boron (B)	Nickel (Ni)	Fluroide (F)	Chloride (Cl)	Nitrate (NO_3-N)	Phosphorous (Total P)	Potassium (K)	SAR
		milligrams per liter													
32-0	12-14-77	.106	.030	.377	.252	.010	<.005	.43	.010	.21	7.1	.33	<.01	-	3.04
32-0	7-13-78	0.30	.023	.052	.288	<.010	<.005	.25	<.010	.20	8.0	.08	<.01	-	2.82
32-0	8-23-78	.072	.030	.078	.168	<.010	<.005	.25	<.010	.20	9.4	.20	.01	-	2.52
32-0	9-29-78	.010	.016	.035	.127	<.010	<.005	.39	<.010	.20	8.5	.11	.02	-	2.44
32-0	11-30-78	.038	<.010	.050	.163	<.010	<.005	.14	<.010	.05	9.9	<.01	<.01	-	5.14+
32-0	2-9-79	.050	.040	.081	.148	<.010	<.005	.42	<.010	.22	9.4	.74	.02	-	2.54
32-0	2-28-79	.030	.020	.082	.094	<.010	<.005	.34	<.010	.20	6.6	<.01	.01	-	2.67
32-0	4-10-79	.036	.020	.043	.130	<.010	<.005	.38	<.010	.23	8.9	.08	<.01	8.4	2.40
32-0	5-14-79	.030	.035	.080	.166	<.010	<.005	.12	<.010	.22	6.1	.69	<.01	9.1	3.15
32-0	6-12-79	.055	.018	.039	.275	<.010	<.005	.10	<.010	.20	8.0	.16	<.01	8.4	3.17
32-0	7-18-79	<.010	.021	.053	.149	<.010	<.005	.29	<.010	.20	9.4	.19	<.01	9.2	2.77
32-0	11-14-79	.018	.021	.079	.051			.33		.24	8.5	.06	.05	5.37	3.13
32-0	2-27-80	.052	.080	.048	.096			.07		.19	6.7	.09	.11	8.10	2.70
32-0	5-27-80	<.020	.030	.036	.101			.16		.16	7.8	.28	.01	7.16	2.28
32-0	9-4-80	.040	.030	.038	.004			.12		.18	7.5	.25	.08	7.76	2.39
32-0	12-10-80	.058	.058	.028	.024			.19		.09	6.6	.07	.01	6.76	1.88
32-R	4-26-77	.118	.010	.018	.077	<.010	.008	.33	.011	.30	5.2	.59	<.01	-	0.90
32-R	7-22-77	.077	.024	.009	.302	<.010	<.005	.45	.012	.24	2.8	.03	.01	-	1.30
32-R	12-14-77	.100	.015	.218	.291	.015	<.005	.57	<.010	.21	2.8	.09	.01	-	0.99
32-R	7-13-78	<.020	.018	.060	.210	<.010	<.005	.57	<.010	.20	9.4	.03	.01	-	1.51
32-R	8-23-78	.014	.015	.128	.216	<.010	<.005	.59	<.010	.20	3.8	.13	.01	-	1.60
32-R	9-29-78	<.010	<.009	.040	.184	<.010	<.005	.67	<.010	.20	3.3	.08	.01	-	1.36
32-R	11-30-78	.026	<.010	.093	.187	<.010	<.005	.42	<.010	.04	2.8	<.01	<.01	-	2.94+
32-R	2-9-79	.050	.030	.136	.150	<.010	<.005	.66	<.010	.18	3.8	.23	.02	-	1.57
32-R	2-28-79	.010	.015	.088	.139	<.010	<.005	.58	<.010	.18	2.4	<.01	.01	-	1.79
32-R	4-10-79	.030	.018	.057	.130	<.010	<.005	.61	<.010	.19	4.7	.06	<.01	6.8	1.50
32-R	5-14-79	.020	.015	.080	.124	<.010	<.005	.42	<.010	.19	2.8	.07	.01	7.4	1.91
32-R	6-12-79	.038	.090	.042	.118	<.010	<.005	.51	<.010	.18	3.8	.10	<.01	7.2	1.71
32-R	7-18-79	.050	.012	.124	.190	<.010	<.005	.54	<.010	.17	4.7	.06	<.01	7.8	1.58
32-R	8-23-79	<.010	<.010	.060	.129	<.010	<.005	.51	<.010	.17	4.7	<.01	.03	7.3	1.41
32-R	11-14-79	<.010	.026	.070	.173			.44		.20	4.6	.12	.06	4.83	1.80
32-R	2-27-80	.048	.062	.048	.095			.41		.17	4.7	.09	.05	7.40	1.31
32-R	5-28-80	.028	.027	.040	.172			.43		.16	4.3	.06	.01	6.44	1.77
32-R	9-4-80	.036	.043	.064	.064			.33		.16	5.8	.26	.09	6.40	1.58
32-R	12-10-80	.048	.047	.031	.083			.40		.09	3.3	.12	.05	6.16	1.44

Table 31. (continued)

Well Number	Date Sampled	Iron (Fe)	Copper (Cu)	Zinc (Zn)	Manganese (Mn)	Lead (Pb)	Cadmium (Cd)	Boron (B)	Nickel (Ni)	Fluroide (F)	Chloride (Cl)	Nitrate (NO_3-N)	Phosphorous (Total P)	Potassium (K)	SAR
		milligrams per liter													
33-0	4-29-77	.130	<.005	.005	<.005	<.010	<.005	.08	.005	.16	9.4	.76	.03	-	1.20
33-0	7-25-77	.074	.026	.030	.009	<.010	<.005	.20	<.010	.31	7.1	.31	.01	-	0.90
33-0	12-15-77	<.010	.020	.155	.030	<.010	<.005	.36	<.010	.21	6.1	.33	.01	-	0.89
33-0	6-22-78	.042	.010	.138	.060	<.010	<.005	.19	<.010	.26	8.0	.22	.02	-	0.93
33-0	7-13-78	.062	.040	.073	.010	<.010	<.005	.25	<.010	.20	8.9	.19	.03	-	1.00
33-0	8-23-78	.040	.010	.730+	.062	<.010	<.005	.27	<.010	.30	7.1	.18	.01	-	1.00
33-0	9-28-78	.048	<.009	.084	.030	<.010	<.005	.40	<.010	.20	10.8	.16	.01	-	0.94
33-0	11-30-78	.020	.028	.197	.024	<.010	<.005	.21	.010	.05	7.1	.12	<.01	-	1.79
33-R	4-29-77	.086	.005	.055	.046	<.010	<.005	.26	.009	.18	9.9	.53	.01	-	1.60
33-R	7-25-77	.078	.018	.115	.200	<.010	<.005	.38	.010	.19	7.5	<.01	.01	-	1.70
33-R	12-15-77	<.010	.018	.238	.133	<.010	<.005	.57	<.010	.17	7.1	.04	.01	-	1.82
33-R	6-22-78	.054	.012	.106	.163	<.010	<.005	.45	<.010	.13	8.5	.11	.02	-	2.11
33-R	7-11-78	<.020	.020	.180	.058	<.010	<.005	.38	<.010	.20	8.0	.19	.01	-	1.83
33-R	8-23-78	.022	.012	.068	.060	<.010	<.005	.37	<.010	.20	8.5	.24	.02	-	1.84
33-R	9-29-78	.058	<.009	.068	.055	<.010	<.005	.48	<.010	.20	9.4	.32	.01	-	1.50
33-R	11-29-78	.100	<.010	.166	.128	<.010	<.005	.18	<.010	.03	8.0	.36	<.01	-	3.86
34-R	4-29-77	.075	.010	.048	.090	.010	.008	.46	.017	.20	11.8	.11	.01	-	2.00
34-R	6-30-77	.190	<.005	.044	.175	.035	<.005	.45	<.010	.16	14.1	.11	.02	-	2.10
34-R	6-30-77	.185	<.005	.043	.165	.030	<.005	.49	<.010	.17	12.7	.21	.01	-	2.10
34-R	7-22-77	.101	.026	.163	.196	<.010	<.005	.52	.015	.20	13.2	.06	<.01	-	2.10
41-0	3-1-79	.025	.032	.016	.050	<.010	<.005	.26	<.010	.20	12.7	.06	.01	-	1.87
41-0	4-10-79	.032	.020	.015	.070	<.010	<.005	.30	<.010	.24	15.1	.04	<.01	8.8	1.50
41-0	5-14-79	.049	.040	.018	.067	<.010	<.005	.11	<.010	.20	12.2	.03	<.01	9.4	1.93
41-0	6-12-79	.040	<.015	.029	.094	<.010	<.005	.14	<.010	.22	11.8	.02	<.01	8.5	1.80
41-0	7-18-79	.055	<.008	.042	.079	<.010	<.005	.23	<.010	.18	13.2	.03	<.01	8.7	1.74
41-0	8-21-79	<.010	<.010	.020	.138	<.010	<.005	.26	<.010	.20	13.6	.04	.03	7.7	1.52
41-0	11-14-79	.020	.042	<.005	.153			.21		.30	13.7	.06	.05	6.45	1.44
41-0	2-27-80	.055	.020	.097	.032			.14		.18	12.9	.03	<.01	7.60	1.31
41-0	5-28-80	.025	.028	.024	.200			.15		.18	9.8	.04	.03	7.16	1.74
41-0	9-4-80	1.280	.045	.166	.144			.09		.20	12.5	.11	.01	7.28	1.63
41-0	12-10-80	.068	.042	.046	.111			.19		.11	11.8	.14	.01	6.34	1.43
41-R	11-30-78	.056	<.010	.092	.434	<.010	<.005	.40	<.010	.04	10.4	.12	<.01	-	2.91
41-R	3-1-79	.024	.024	.215	.298	<.010	<.005	.48	<.010	.18	8.9	.01	.01	-	1.83
41-R	4-10-79	.036	<.010	.233	.213	.010	.012	.46	<.010	.23	9.9	.06	<.01	6.5	1.50
41-R	5-14-79	.020	.022	.131	.133	<.010	<.005	.37	<.010	.23	7.5	.11	<.01	7.1	1.91
41-R	6-12-79	.036	<.015	.083	.171	<.010	<.005	.28	<.010	.20	8.0	.01	<.01	6.6	1.75

Table 31. (continued)

Well Number	Date Sampled	Iron (Fe)	Copper (Cu)	Zinc (Zn)	Manganese (Mn)	Lead (Pb)	Cadmium (Cd)	Boron (B)	Nickel (Ni)	Fluroide (F)	Chloride (Cl)	Nitrate (NO_3-N)	Phosphorous (Total P)	Potassium (K)	SAR
		milligrams per liter													
41-R	7-18-79	.055	<.008	.148	.163	<.010	<.005	.30	<.010	.21	9.9	.03	<.01	6.8	1.74
41-R	8-21-79	<.010	.016	.286	.106	<.010	<.005	.29	<.010	.22	9.9	.07	.05	6.6	1.60
41-R	11-14-79	<.010	.011	.153	.138			.26		.28	11.7	.05	.06	4.95	1.73
41-R	5-28-80	.095	.023	.160	.149			.23		.21	7.6	.07	-	6.14	1.76
41-R	9-4-80	.256	.030	.172	.046			.20		.20	8.6	.08	.07	6.96	1.93
41-R	12-10-80	.258	.036	.080	.070			.26		.12	7.5	.15	.01	5.92	1.53
41-M	3-1-79	.023	.026	.010	<.010	<.010	<.005	.17	<.010	.24	4.2	.13	<.01	-	5.12
41-M	5-14-79	.036	.010	.010	<.010	<.010	<.005	.14	<.010	.42	2.8	.05	.03	8.7	16.80
41-M	6-12-79	.122	<.015	.016	.010	<.010	<.005	.17	<.010	.42	3.8	.04	<.01	7.4	12.30
41-M	7-18-79	.075	<.008	.011	.036	<.010	<.005	.24	<.010	.40	5.2	.01	.03	7.4	9.14
41-M	8-21-79	<.010	<.010	.038	.005	<.010	<.005	.21	<.010	.37	3.8	.03	.04	7.1	6.33
41-M	11-14-79	.015	.035	<.005	.078			.13		.41	3.9	.06	.11	5.00	10.90
41-M	2-27-80	.067	.023	.029	<.010			.23		.32	3.8	.03	.11	7.40	11.99
41-M	5-28-80	.025	.022	<.010	.100			.26		.34	4.1	.07	.06	6.00	8.71
41-M	9-4-80	.080	.030	.080	.004			.22		.32	4.1	.05	.08	6.38	8.92
42-0	4-11-79	.102	.015	.015	.240	<.010	<.005	.30	<.010	.42	4.2	.01	<.01	4.6	0.70
42-0	5-15-79	.020	.018	.014	.255	<.010	<.005	-	<.010	.43	3.3	.03	<.01	5.4	0.94
42-0	6-12-79	.030	<.015	.215	.188	<.010	<.005	.11	<.010	.31	3.8	.02	<.01	3.5	0.74
42-0	7-18-79	.050	<.008	.044	.370	<.010	<.005	.18	<.010	.32	4.7	.01	<.01	4.7	0.67
42-0	8-21-79	<.010	.020	.044	.219	<.010	<.005	.15	<.010	.29	3.8	<.01	.03	4.2	0.72
42-0	11-14-79	.021	.018	.010	.273			.09		.40	4.0	.07	.05	5.60	0.67
42-0	2-27-80	.052	.036	.177	.406			.11		.29	3.5	.05	.03	3.10	0.54
42-0	5-28-80														
42-0	12-10-80	.055	.035	.031	.197			.16		.17	2.4	.10	<0.01	2.80	0.65
42-R	4-11-79	.030	.018	.028	.270	<.010	<.005	.47	<.010	.18	4.2	<.01	.01	5.8	1.20
42-R	5-15-79	.020	.030	.023	.224	<.010	<.005	.28	<.010	.14	2.8	.02	<.01	6.1	1.58
42-R	6-12-79	.030	<.015	.147	.242	<.010	<.005	.39	<.010	.10	3.3	.01	<.01	4.8	1.34
42-R	7-18-79	.115	<.008	.101	.325	<.010	<.005	.30	<.010	.11	3.3	.02	.04	6.0	1.29
42-R	8-21-79	<.010	<.010	.297	.237	.018	<.005	.31	<.010	.14	3.3	.03	.03	5.1	1.26
42-R	11-14-79	.128	.015	.013	.293			.19		.24	4.0	<.01	.07	4.44	0.96
42-R	2-27-80	.067	.066	.100	.143			.17		.21	2.6	.05	.06	4.00	0.76
42-R	5-28-80	.032	.014	.084	.222			.25		.21	3.4	.06	.02	4.34	1.17
42-R	9-4-80	.056	.038	.106	.106			.17		.24	4.4	.06	.09	4.22	1.09
42-R	12-10-80	.037	.039	.176	.122			.20		.13	1.4	.11	<0.01	3.80	0.84
42-M	4-11-79	.024	<.010	.090	.100	<.010	<.005	.40	<.010	.18	2.8	.03	.01	6.8	3.20
42-M	5-15-79	.020	.027	.098	.108	<.010	<.005	.22	<.010	.17	2.4	.01	<.01	7.2	4.15
42-M	6-12-79	.036	<.015	.110	.070	<.010	<.005	.22	<.010	.16	2.8	.02	<.01	6.8	4.07

Table 31. (continued)

Well Number	Date Sampled	Iron (Fe)	Copper (Cu)	Zinc (Zn)	Manganese (Mn)	Lead (Pb)	Cadmium (Cd)	Boron (B)	Nickel (Ni)	Fluroide (F)	Chloride (Cl)	Nitrate (NO_3-N)	Phosphorous (Total P)	Potassium (K)	SAR
		milligrams per liter													
42-M	7-18-79	.047	<.008	.188	.163	<.010	<.005	.28	<.010	.15	3.8	<.01	.01	7.6	3.75
42-M	8-21-79	<.010	.020	.275	.103	<.010	<.005	.22	<.010	.16	3.3	.02	.03	6.8	4.10
42-M	11-14-79	<.010	.020	.148	.145			.18		.20	2.9	.13	.07	7.40	4.22
42-M	2-27-80	.052	.021	.260	.083			.26		.15	3.3	.06	.03	6.50	3.23
42-M	5-28-80	.027	.018	.086	.164			.29		.14	2.6	.05	-	6.62	4.25
42-M	9-4-80	.072	.025	.138	.060			.25		.14	3.0	.07	.06	6.66	3.96
42-M	12-10-80	.026	.038	.182	.092			.27		.08	2.8	.07	.02	5.76	3.56
43-0	3-1-79	.015	.034	.073	.275	<.010	<.005	.58	<.010	.16	7.1	.01	.01	-	1.67
43-0	4-10-79	.120	<.010	.050	.210	<.010	<.005	.59	<.010	.18	7.5	<.01	.01	8.0	1.40
43-0	5-15-79	.024	.020	.042	.085	<.010	<.005	.21	<.010	.31	10.4	.05	<.01	5.8	1.53
43-0	6-12-79	.120	<.015	.082	.152	<.010	<.005	.27	<.010	.17	6.1	<.01	<.01	6.8	1.81
43-0	7-18-79	.060	.026	.091	.179	<.010	<.005	.25	<.010	.18	7.5	.04	.01	7.3	1.48
43-0	11-14-79	.032	.013	.029	.220			.37		.24	7.6	<.01	.06	10.42	1.29
43-0	2-27-80	.069	.020	.042	.048			.15		.19	6.9	.04	.03	5.80	1.03
43-0	5-28-80	.030	.015	.030	.189			.24		.20	7.4	.05	-	5.66	1.26
43-0	9-4-80	.080	.035	.064	.066			.18		.20	9.6	<0.01	.05	5.32	1.18
43-0	12-9-80	.189	.066	.049	.096			.19		.12	6.1	.08	<0.01	4.66	0.98
43-R	3-1-79	.076	.030	.060	.250	<.010	<.005	.37	<.010	.19	3.8	.09	.01	-	1.49
43-R	4-10-79	.060	.018	.088	.364	<.010	<.005	.62	<.010	.19	7.1	.15	.01	8.4	1.70
43-R	5-15-79	.028	.024	.500	.215	<.010	<.005	.45	<.010	.16	6.1	.01	.01	8.4	2.30
43-R	6-12-79	.045	<.015	.310	.341	<.010	<.005	.53	<.010	.16	6.6	<.01	<.01	7.1	2.11
43-R	7-18-79	.061	.016	.356	.284	<.010	<.005	.48	<.010	.13	5.6	<.01	.04	8.1	2.03
43-R	11-14-79	.136	.011	.005	.350			.57		.18	4.5	.04	.06	6.75	1.93
43-R	2-27-80	.170	.010	.029	.190			.37		.13	5.9	.05	.03	7.00	1.72
43-R	5-28-80	.150	.018	.010	.380			.48		.13	5.7	.03	-	6.90	2.04
43-R	9-4-80	.188	.028	.046	.260			.36		.14	6.8	.05	.07	7.04	2.12
43-R	12-9-80	.221	.048	.055	.282			.50		.07	3.3	.08	.02	5.88	1.86
43-M	3-1-79	.030	.028	.370	.201	<.010	<.005	.54	<.010	.24	5.2	.14	.02	-	2.55
43-M	4-10-79	.046	<.010	.050	.210	<.010	<.005	.63	<.010	.23	7.5	.05	.01	10.4	2.60
43-M	5-15-79	.039	.014	.498	.174	<.010	<.005	.58	<.010	.18	4.7	<.01	.01	8.8	2.75
43-M	6-12-79	.082	<.015	.360	.212	<.010	<.005	.53	<.010	.15	6.6	<.01	<.01	7.8	2.50
43-M	7-18-79	.058	<.008	.905	.180	<.010	<.005	.46	<.010	.15	6.1	.09	<.01	9.2	2.58
43-M	11-14-79	.010	.016	.343	.290			.51		.18	6.7	<.01	.05	11.20	2.40
43-M	2-27-80	.090	.015	.176	.037			.36		.13	6.0	.18	.02	10.60	1.89
43-M	5-28-80	.025	.040	.384	.290			.49		.13	5.0	.08	.01	7.52	2.49
43-M	9-4-80	.196	.022	.142	.180			.37		.12	6.2	.05	.07	7.36	2.57
43-M	12-9-80	.284	.053	.226	.202			.41		.07	2.8	.08	.01	6.38	2.17

Table 31. (continued)

Well Number	Date Sampled	Iron (Fe)	Copper (Cu)	Zinc (Zn)	Manganese (Mn)	Lead (Pb)	Cadmium (Cd)	Boron (B)	Nickel (Ni)	Fluroide (F)	Chloride (Cl)	Nitrate (NO_3-N)	Phosphorous (Total P)	Potassium (K)	SAR
		milligrams per liter													
CM-1	2-9-79	.400	.025	1.098	.388	<.010	<.005	.59	<.010	.23	6.6	.12	.02	-	2.48
CM-1	2-28-79	.100	.020	.049	.366	<.010	<.005	.47	<.010	.19	6.1	.01	.06	-	2.40
CM-1	4-10-79	.048	.030	.071	.402	<.010	<.005	.48	<.010	.20	5.2	<.01	<.01	8.6	1.90
CM-1	5-13-79	.148	<.010	.251	.305	<.010	<.005	.53	<.010	.18	5.6	<.01	.01	9.6	2.27
CM-1	6-13-79	.050	.026	.030	.508	<.010	<.005	.32	<.010	.18	4.2	.03	<.01	8.3	2.16
CM-1	7-18-79	.520	.024	.014	.550	<.010	<.005	.47	<.010	.17	7.5	.02	.01	9.1	2.19
CM-1	8-21-79	1.084	<.010	.043	.371	<.010	<.005	.48	<.010	.18	6.6	.01	.03	8.2	1.94
CM-1	11-14-89	.010	.014	.161	.247			.44		.20	6.4	.05	-	12.55	2.12
CM-1	2-27-80	.059	.034	.031	.308			.32		.18	6.0	.30	.19	8.00	1.48
CM-1	5-27-80	.850	.138	.202	.670			.35		.19	5.0	.10	.04	7.10	2.12
CM-1	9-4-80	.192	.051	.040	.684			.32		.20	5.2	.07	.02	7.42	2.04
CM-1	12-9-80	.274	.040	.134	.583			.30		.10	3.3	.25	<.01	6.84	2.04
CR-2	11-30-78	.044	.027	.080	.710	<.010	<.005	.32	<.010	.09	49.9	.09	.02	-	7.13+
CR-2	2-9-79	.080	.056	.712	.340	<.010	<.005	.57	<.010	.22	51.8	.30	.01	-	3.17
CR-2	2-28-79	.040	.035	.578	.559	<.010	<.005	.40	<.010	.20	49.4	<.01	.04	-	3.34
CR-2	4-10-79	.040	.028	.216	.465	<.010	.020	.38	<.010	.24	50.8	.22	<.01	14.3	3.40
CR-2	5-13-79	.043	.075	.143	.365	<.010	<.005	.47	<.010	.21	43.3	.13	<.01	14.4	3.16
CR-2	6-13-79	.020	.040	.265	.455	<.010	<.005	.33	<.010	.21	44.7	.08	<.01	13.3	3.51
CR-2	7-18-79	.075	.020	.133	.272	<.010	<.005	.33	<.010	.20	45.2	.03	<.01	13.6	3.38
CR-2	2-28-80	<.010	.028	.427	.207			.19		.29	40.9	.23	.05	13.30	3.18
CR-2	5-27-80	.060	.068	.360	.404			.25		.21	49.4	.16	.05	12.40	3.08
CR-2	9-4-80	.076	.048	.336	.312			.22		.24	48.7	.25	.03	13.04	3.30
CR-2	12-9-80	.126	.058	.835	.391			.20		.13	49.4	.14	<.01	12.36	3.07
D-2	7-14-78	.090	.020	.024	.378	<.010	<.005	.50	<.010	.20	16.5	<.01	.01	-	2.06
D-2	8-22-78	.028	.024	.022	.324	<.010	<.005	.46	<.010	.20	14.1	<.01	.01	-	1.90
D-2	9-28-78	.030	.012	.026	.376	<.010	<.005	.43	.010	.20	16.0	.01	.01	-	1.75
ODS*-1	4-17-77	.040	.005	.019	.034	.018	<.005	.18	.013	.18	2.4	.30	<.01	-	0.50
OBD**	4-20-77	.020	<.005	.005	.029	<.01	.012	.16	.005	.20	.5	.21	.34	-	0.40

*Dip sample from overburden blast hole approximately 90 m NE of Well 29-R

**Spring discharge from highwall 30 cm above top of coal seam

+Chemical value not considered representative and therefore not utilized in determining range and means

APPENDIX C

LABORATORY PROCEDURES AND QUALITY CONTROL

Overburden materials in this study were characterized in two different phases. Samples from initial reconnaissance of 38 boreholes on a 305 m grid were all analyzed by Montana Testing Laboratories located in Great Falls, MT. Procedures used were those recommended by the Montana Department of State Lands for overburden analysis (MDSL, 1977), and are summarized in Table 32. These 38 boreholes were drilled by Teton Exploration Drilling Company, Inc. of Casper, Wyoming. A portadrill was used and the drilling method was air-rotary without the use of solutions. Drill stem lubricants used had metal additives (Zn, Cu or Pb) and caused some contamination of samples regarding these element.

Following this initial reconnaissance, the same overburden area was sampled more intensively (60 m grid) with a Bucyrus-Erie 60-R drilling platform. This equipment, owned by the Western Energy Company of Colstrip, MT was used to prepare overburden blast holes. All these samples were analyzed in the Field Laboratory operated by the Montana Agricultural Experiment Station Reclamation Research Unit. The Field Laboratory used in this investigation was housed in a three trailer complex located adjacent to the study area near Colstrip, MT. Overburden sampling and lab work often preceded the dragline by only a few days, so rapid analysis of materials was essential for planning overburden handling demonstrations. Therefore some analytical procedures were modified to expedite lab work.

Most lab procedures recommend air-drying soil material before chemical analysis, rather than oven drying. Rapid oven drying may rupture soil constituents and result in misrepresentative chemical analyses. Since the experimental design required rapid sample turn around times, a procedure was established which reduced the moisture content of samples to approximately an air-dry state using a forced air oven set at 80°C and a drying time of 8 hours. One-hundred overburden samples were split so half could be air-dried for 5 days and half oven-dried. Linear regression coefficients (r) between air and oven drying for electrical conductance, Pb and Ni were excellent (.98, .96, .93 respectively), indicating little or no difference existed between the two drying procedures. Regression coefficients for pH (.82) and Zn (.67) were not as good, suggesting some error was introduced into the analysis due to oven drying. The oven drying procedure was implemented into this study and it was expected that analytical errors, if any, would not affect the study plan.

All oven-dried overburden samples in the Field Laboratory were fragmented and passed through a 1 mm stainless steel sieve.

Determination of Pb, Ni, Cu and Zn in the Field Laboratory was performed with the DTPA-TEA extraction method (Norwell and Lindsay, 1972), except the soil and extraction agent was shaken in 50 ml beakers for 2 hours at 180 opm (Soltanpour et al., 1976). Sample splits for quality control on trace element

Table 32. Overburden laboratory analysis procedures.

Sample Preparation

U.S.D.A. Agriculture Handbook No. 60, 1954. Diagnosis and Improvement of Saline and Alkali Soils, p. 83-84. All analyses conducted on samples passed through at least a 20-mesh sieve (0.84 mm).

Sodium Adsorption Ratio (SAR)

This determination was made on the extract from a water saturated soil paste. Agriculture Handbook No. 60, United States Dept. of Agriculture, 1954, p. 84-88.

$\text{SAR} = \text{meq/L Na} / [\text{meq/L Ca} + \text{meq/L Mg})/2]^{1/2}$

Calcium, Magnesium, Sodium (water extract)

Determination by Atomic Absorption Spectroacopy using saturation extract. Agriculture Handbook No. 60, United States Dept. of Agriculture, 1954, p. 84-88.

Calcium, Magnesium, Sodium, Potassium (NH_4OAc extract)

Agriculture Handbook No. 60, USDA, 1954, Method 18, p. 100-101.

Conductance

This determination was made on the extract from a water saturated soil paste. Agriculture Handbook No. 60, USDA, 1954, p. 89.

pH

This determination was made on a water saturated soil paste. Agriculture Handbook No 60. USDA, 1954, p. 102.

Saturation Percentage

Grams H_2O/grams soil to attain saturation times 100. Agriculture Handbook No. 60, USDA, 1954, method 2 and 3a.

Particle Size Analysis (hydrometer)

A.S.A. Agronomy Monograph No. 9, 1965. Methods of Soil Analysis. Part 1. Method 43-5, p. 562-566. Textural triangle was used for texture class identification, Soil Survey Manual, 1951, USDA, p. 209.

Nitrate

Phenoldisulfonic acid method. A.S.A. Monograph No. 9, 1965. Methods of Soil Analysis, Part 2, p. 1212-1214.

Boron

Hot water extraction. A.S.A. Agronomy Monograph No. 9, 1965. Methods of Soil Analysis. Part 2. Method 75-4, p. 1062-1063.

Copper, Cadmium, Iron, Lead, Manganese, Zinc, and Nickel

Atomic absorption analysis of DTPA extract. Soil Science Soc. of American Proceeding, Vol. 35, No 4, 1971, p. 600-602.

Molybdenum

Acid ammonium oxalate extraction. A.S.A. Agronomy Monograph No. 9, 1965. Methods of Soil Analysis. Part 2, p. 1054-1057.

Selenium

Determination by the gaseous hydride method after hot water extraction. The Se is reduced from +6 oxidation state to +4 oxidation state by addition of $NaBH_4$ solution to produce H and convert Se to SeH_2. The gaseous SeH_2 is swept into N flame of AA for determination. EPA, Methods for Chemical Analysis of Water and Waste, 1974. Extraction procedure also presented in A.S.A. Monograph No. 9, 1965. Methods of Soil Analysis. Part 2, 80-3.2.2, p. 1122.

Mercury

Acid extraction determined by cold vapor generation and atomic absorption spectroacopy. Anal. Chem. vol. 40, No. 114 (1968). Also Manual of Methods for Chemical Analysis of Water and Wastes, E.P.A., 1974, p. 137, Procedure 8.1.

Ammonium

Extraction with NaCL, distillation and titration. A.S.A. Agronomy Monograph No. 9. Methods of Soil Analysis. Part 2, p. 1185-1212.

Phosphorus

Colorimetric determination of $NaHCO_3$ extraction. A.S.A. Monograph No. 9. Methoda of Soil Analysis, Part 2, Method 73-4.4, p. 1044-1047.

Exchangeable Sodium Percentage (ESP)

Exchangeable Na determined by NH_4OAc extraction and subtraction of the H_2O soluble fraction. Cation exchange capacity (CEC) determined by NaOAc saturation and NH_4OAc extraction. Agriculture Handbook No. 60, USDA, 1954, Methods 18 and 19, p. 100-101. $\text{ESP} = [(\text{meq/100g Na})/(\text{meq/100g CEC})]\ 100$

analysis were performed every 6 months with the Trace Element Laboratory at Montana State University[1] and with Northern Testing Laboratories[2]. Comparison between these labs consistently resulted in linear correlation coefficients (r) >0.9, and generally near 1.0. Slopes were near 1.0 for Pb and Ni, while Zn values were generally higher (e.g. 1.0 vs. 1.5 ppm) at Northern Testing Laboratories and the Trace Element Laboratory compared to the Field Laboratory. Sample concentration reproducibility at the Field Lab was very good with both correlation coefficients and slopes approaching 1.0 for Pb and Ni, while these statistical values were approximately 0.9 for Zn.

Overburden electrical conductivity (EC) and pH should be performed on the extract from a water saturated soil paste (USDA, 1969). This procedure is laborious and requires a 24 hour sample turn around time. In order to expedite this work in the Field Laboratory, EC and pH were determined on the extract from a 1:1 overburden/water mixture (USDA, 1969). Each 1:1 mixture was shaken for 30 minutes at 140 oscillations/minute, then samples were remixed and allowed to settle for 20 minutes, and gravity filtered through a Whatman No. 40 paper filter. Approximately 150 overburden samples were split and analyzed for EC and pH by the saturated extract and 1:1 extract procedures. Linear correlation coefficients (r) between the two procedures were excellent for EC (.97) and fair for pH (.77). The standard deviation for EC comparison was ±.02 mmhos/cm and ±.07 for pH. Therefore, even though r was only fair for the comparison in pH, the 1:1 extract produced pH values that were ±.14 units from the pH of the saturation extract in 95% (2 std. dev.) of all samples analyzed. This was acceptable accuracy for this overburden study.

Overburden particle size distribution was determined for the sand, silt and clay fractions which were generalized into the size categories of 2-.05 mm, .05-.002 mm, and <.002 mm, respectively (Day, 1965). The procedure of Day was modified in the Field Laboratory by separating the sand and silt fractions at the 40 second hydrometer reading, and the silt and clay fraction at the 2 hour reading. Hydrometer readings at these times would, in effect, underestimate silt content and overestimate clay content by a few percent. This loss in accuracy was accepted in order to expedite sample analysis. The standard Day procedure required silt-clay separation with about an 8 hour hydrometer reading in the sedimentation cylinder.

1/ Trace Element Laboratory, Reclamation Research Program, Montana State University, Bozeman. Dr. F.F. Munshower, supervisor.

2/ Northern Testing Laboratories, Billings, MT. Mr. J. Standish, supervisor.

APPENDIX D

SPOIL CHEMISTRY AS A FUNCTION OF TIME FOR DRILL SITES S-4 THROUGH S-19 IN AREA B OF THE WESTERN ENERGY COMPANY MINE NEAR COLSTRIP, MT.

Table 33. Spoil chemistry as a function of time for drill site S-4 in Area B of the Western Energy Company Mine near Colstrip, MT.

Spoil Depth(m)	pH	EC mmhos/cm	Pb	Ni	Zn	Sand	Silt	Clay
			ppm			%		
			February, 1978					
0-.9	7.4	2.4	3.7	0.1	1.0	62.6	18	19.4
.9-1.8	7.5	2.3	4.9	0.5	1.8	40.6	24	35.4
1.8-2.7	7.5	2.7	2.9	0.5	1.1	54.6	18	27.4
2.7-3.7	7.3	4.9	1.5	0.4	0.9	42.6	34	23.4
3.7-4.6	7.1	4.1	1.1	0.1	0.6	76.6	18	5.4
4.6-5.5	7.0	6.3	0.9	0.1	0.5	56.6	26	17.4
5.5-6.4	7.4	3.6	43.	0.7	2.9	42.6	32	25.4
6.4-7.3	7.2	2.8	12.	0.1	0.9	84.6	12	3.4
7.3-8.2	7.2	2.9	9.2	0.3	0.9	84.6	12	3.4
8.2-9.1	7.3	3.1	24.	0.3	1.2	66.6	24	9.4
9.1-10.0	7.4	3.4	5.7	0.1	0.6	76.6	18	5.4
10.0-11.0	7.2	4.5	7.2	0.3	0.7	68.6	20	11.4
11.0-11.9	7.4	2.7	10.2	0.6	5.2	64.6	26	9.4
11.9-12.8	7.4	3.2	4.1	0.6	1.2	59.4	26	14.6
12.8-13.7	7.5	2.3	5.4	0.7	1.2	65.4	18	16.6
13.7-14.6	7.4	3.0	4.0	0.9	1.4	58.6	20	21.4
Mean	7.3	3.4	8.7	0.4	1.4	62.8	21.6	15.6
Std. Dev.	0.1	1.1	10.7	0.2	1.2	13.7	6.2	9.5
			October, 1978					
0-.9	7.0	2.8	3.2	1.1	3.4	38.6	30	31.4
.9-1.8	7.0	4.9	1.9	0.6	2.1	40.6	30	29.4
1.8-2.7	7.0	5.3	1.2	0.5	1.1	46.6	26	27.4
2.7-3.7	7.0	6.0	1.2	0.5	1.3	34.6	36	29.4
3.7-4.6	7.0	5.9	1.2	0.4	1.2	28.6	34	37.4
4.6-5.5	7.1	5.9	1.3	0.5	1.2	44.6	26	29.4
5.5-6.4	7.1	6.0	1.6	0.6	1.3	40.6	28	31.4
6.4-7.3	7.1	5.7	1.6	0.8	1.2	34.6	34	31.4
7.3-8.2	7.1	4.1	2.3	0.8	1.4	36.6	32	31.4
8.2-9.1	7.1	6.0	4.7	0.5	1.3	26.6	38	35.4
9.1-10.0	7.2	5.0	2.1	0.9	1.5	34.6	32	33.4
10.0-11.0	7.2	3.2	2.5	0.9	1.5	32.6	42	25.4
11.0-11.9	7.3	3.0	6.5	1.4	3.3	20.6	28	51.4
11.9-12.8	7.4	3.0	5.3	1.6	2.0	16.6	38	45.4
12.8-13.7	7.4	1.7	2.7	1.1	1.0	18.6	34	47.4
13.7-14.6	7.4	3.9	5.8	2.2	1.1	42.6	30	27.4
Mean	7.2	4.5	2.6	0.9	1.6	33.6	32.4	34.0
Std. Dev.	0.2	1.4	1.7	0.5	0.7	9.2	4.6	7.6
			April, 1979					
0-.9	6.8	2.3	1.6	0.6	1.7			
.9-1.8	7.0	4.1	2.2	0.8	2.3			
1.8-2.7	7.4	4.6	1.3	0.3	0.9			
2.7-3.7	6.9	3.6	1.3	0.4	0.8			
3.7-4.6	6.8	4.9	1.5	0.6	1.2			
4.6-5.5	6.9	1.8	3.7	1.6	1.3			
5.5-6.4	7.4	2.5	1.9	0.6	2.0			
6.4-7.3	7.4	2.5	1.9	0.6	2.0			
7.3-8.2	7.4	2.1	2.5	0.9	4.7			
8.2-9.1	7.8	2.4	2.0	0.7	4.9			
9.1-10.0	7.8	2.2	0.9	0.2	1.0			
10.0-11.0	7.9	2.4	0.8	0.3	0.5			
11.0-11.9	7.9	2.4	0.8	0.3	0.5			
11.9-12.8	7.9	2.4	0.8	0.3	0.5			
12.8-13.7	7.9	2.4	0.8	0.3	0.5			
13.7-14.6	7.8	2.1	0.9	0.4	0.6			
Mean	7.4	2.8	1.6	0.6	1.6			
Std. Dev.	0.4	1.0	0.8	0.3	1.4			

Table 34. Spoil chemistry as a function of time for drill site S-6 in Area B of the Western Energy Company mine near Colstrip, MT.

Spoil		EC	Pb	Ni	Zn	Sand	Silt	Clay
Depth(m)	pH	mmhos/cm		ppm			%	
			February, 1978					
0-.9	7.3	2.3	2.4	0.2	1.0	74.6	16	9.4
.9-1.8	7.3	4.6	6.6	0.3	1.6	48.6	26	25.4
1.8-2.7	7.5	1.9	2.6	0.9	1.5	36.6	30	33.4
2.7-3.7	7.4	4.3	2.7	0.6	1.1	28.6	36	35.4
3.7-4.6	7.3	3.3	0.8	0.1	0.4	78.6	16	5.4
4.6-5.5	7.4	3.2	0.3	0.0	0.6	72.6	18	9.4
5.5-6.4	7.3	5.6	4.0	0.4	1.0	50.6	28	21.4
6.4-7.3	7.4	3.6	2.5	0.7	1.0	58.6	20	21.4
7.3-8.2	7.2	3.9	0.8	0.1	0.5	78.6	12	9.4
8.2-9.1	7.2	4.0	0.7	0.1	0.6	80.6	12	7.4
9.1-10.0	7.4	2.1	1.3	0.0	0.5	76.6	14	9.4
10.0-11.0	7.2	2.4	1.3	0.6	1.0	66.6	20	13.4
11.0-11.9	7.1	3.6	2.4	0.1	0.6	78.6	14	7.4
11.9-12.8	7.3	2.3	7.5	0.6	1.0	62.6	26	11.4
Mean	7.3	3.4	2.6	0.3	0.9	63.1	20.6	15.7
Std. Dev.	0.1	1.1	2.2	0.3	0.4	16.9	7.5	9.9
			October, 1978					
0-.9	7.3	5.1	1.5	0.4	1.5	36.6	32	31.4
.9-1.8	7.3	4.2	1.3	0.3	1.1	36.6	28	35.4
1.8-2.7	7.4	3.9	1.1	0.1	0.5	56.6	16	27.4
2.7-3.7	7.3	3.8	1.1	0.2	0.4	34.6	32	33.4
3.7-4.6	7.2	4.3	1.1	0.2	0.6	46.6	26	27.4
4.6-5.5	7.3	4.5	1.3	0.5	0.8	46.6	22	31.4
5.5-6.4	7.4	2.4	2.0	0.6	0.9	60.6	14	25.4
6.4-7.3	7.4	1.8	1.9	0.6	2.0	56.6	18	25.4
7.3-8.2	7.4	1.7	1.5	0.4	0.6	64.6	16	19.4
8.2-9.1	7.2	3.5	1.2	0.4	0.7	46.6	30	23.4
9.1-10.0	7.4	3.1	2.1	0.6	1.8	34.6	38	27.4
10.0-11.0	7.4	2.5	2.7	1.2	1.3	34.6	38	27.4
11.0-11.9	7.5	2.0	5.6	1.0	3.0	54.6	18	27.4
11.9-12.8	7.5	1.9	2.1	1.0	1.0	20.6	32	47.4
Mean	7.4	3.2	1.9	0.5	1.2	45.0	25.7	30.2
Std. Dev.	0.1	1.1	1.2	0.3	0.7	12.6	8.3	4.4
			April, 1979					
0-.9	7.4	3.4	2.7	1.0	5.8			
.9-1.8	7.8	1.4	0.9	0.2	0.6			
1.8-2.7	7.4	4.9	1.0	0.0	0.4			
2.7-3.7	7.3	5.8	0.8	0.2	0.7			
3.7-4.6		6.5	0.9	0.3	1.0			
4.6-5.5	7.0	5.1	1.0	0.4	2.9			
5.5-6.4	7.0	5.1	1.0	0.4	2.9			
6.4-7.3	7.5	4.8	0.7	0.2	1.5			
7.3-8.2	7.5	4.8	0.7	0.2	1.5			
8.2-9.1	7.9	3.7	0.6	0.1	0.4			
9.1-10.0	6.9	5.6	0.9	0.3	0.6			
10.0-11.0	7.3	5.7	0.8	0.2	0.5			
11.0-11.9	7.5	3.8	1.5	0.8	1.9			
11.9-12.8	7.4	2.4	2.2	1.1	1.1			
Mean	7.4	4.5	1.1	0.4	1.6			
Std. Dev.	0.3	1.4	0.6	0.3	1.5			

Table 35. Spoil chemistry as a function of time for drill site S-7 in Area B of the Western Energy Company mine near Colstrip, MT.

Spoil Depth(m)	pH	EC mmhos/cm	Pb	Ni	Zn	Sand	Silt	Clay
				ppm			%	
				February, 1978				
0-.9	7.4	2.2	4.0	1.0	3.7	28.6	32	39.4
.9-1.8	7.5	1.8	6.0	0.8	4.7	28.6	32	39.4
1.8-2.7	7.6	1.7	3.9	0.7	2.5	26.6	38	35.4
2.7-3.7	7.5	2.8	5.6	0.7	3.1	50.6	16	33.4
3.7-4.6	7.1	5.9	51.	0.3	3.2	44.6	36	19.4
4.6-5.5	7.1	5.9	51.	0.3	3.2	44.6	36	19.4
5.5-6.4	7.3	3.2	4.3	0.5	1.2	44.6	28	27.4
6.4-7.3	7.4	2.2	4.9	1.2	1.4	32.6	32	35.4
7.3-8.2	7.4	2.1	5.8	1.1	1.1	38.6	32	29.4
8.2-9.1	7.4	3.0	2.2	0.8	0.9	36	34	30
9.1-10.0	7.1	5.2	1.6	0.5	0.7	48	26	26
10.0-11.0	7.2	3.8	1.0	0.3	0.5	70.6	16	13.4
11.0-11.9	7.2	3.7	2.2	0.4	0.9	64.6	20	15.4
11.9-12.8	7.0	5.4	1.1	0.4	0.8	58.6	24	17.4
12.8-13.7	7.0	5.8	1.7	0.3	0.7	48.6	28	23.4
13.7-14.6	7.0	6.2	1.1	0.4	0.9	42.6	34	23.4
14.6-15.5	7.0	6.0	1.0	0.3	0.9	40.6	34	25.4
Mean	7.2	3.9	8.7	0.6	1.8	44.1	29.3	26.6
Std. Dev.	0.2	1.7	16.0	0.3	1.3	12.4	6.8	8.1
				October, 1978				
0-.9	7.4	1.5	1.6	0.4	1.3	62.6	18	19.4
.9-1.8	7.4	4.2	1.2	0.3	0.5	68.6	14	17.4
1.8-2.7	7.3	4.5	1.2	0.3	0.6	60.6	20	19.4
2.7-3.7	7.3	5.1	1.5	0.4	0.9	60.6	20	19.4
3.7-4.6	7.3	5.4	1.1	0.4	0.9	52.6	22	25.4
4.6-5.5	7.3	4.7	1.8	0.7	0.9	40.6	32	27.4
5.5-6.4	7.3	2.8	2.5	1.0	1.4	46.6	26	27.4
6.4-7.3	7.4	2.2	2.1	0.7	1.1	50.6	22	27.4
7.3-8.2	7.5	2.1	4.3	1.4	2.3	50.6	22	27.4
8.2-9.1	7.6	2.5	2.1	0.7	1.2	36.6	22	41.4
9.1-10.0	7.6	2.4	0.9	0.2	0.4	68.6	12	19.4
10.0-11.0	7.6	4.0	0.9	0.3	0.6	56.6	20	23.4
11.0-11.9	7.6	3.2	1.2	0.4	0.6	40.6	32	27.4
11.9-12.8	7.4	4.0	1.1	0.3	0.6	52.6	24	23.4
12.8-13.7	7.5	2.5	0.8	0.2	0.4	62.6	16	21.4
13.7-14.6	7.4	3.0	0.9	0.3	0.4	68.6	14	17.4
14.6-15.5	7.5	2.7	0.7	0.3	0.3	70.6	10	19.4
Mean	7.4	3.3	1.5	0.5	0.8	55.9	20.4	23.8
Std. Dev.	0.1	1.2	0.9	0.3	0.5	10.7	6.2	5.9
				April, 1979				
0-.9	7.0	1.6	0.9	0.2	1.9			
.9-1.8	7.0	3.3	1.1	0.3	1.2			
1.8-2.7	7.4	1.7	0.8	0.3	0.4			
2.7-3.7	6.9	3.4	0.9	0.1	0.5			
3.7-4.6	7.0	4.1	1.1	0.5	1.7			
4.6-5.5	7.1	2.3	0.9	0.1	0.6			
5.5-6.4	7.1	1.0	0.7	0.1	0.2			
6.4-7.3	7.7	3.7	0.8	0.1	0.7			
7.3-8.2	7.1	2.5	1.1	0.4	1.4			
8.2-9.1	7.1	4.8	0.8	0.1	0.7			
9.1-10.0	6.8	4.1	1.2	0.5	0.7			
10.0-11.0	7.9	2.7	1.2	0.4	0.8			
11.0-11.9	7.9	2.7	1.2	0.4	0.8			
11.9-12.8	7.9	2.5	1.3	0.5	4.2			
12.8-13.7	7.9	2.5	1.3	0.5	4.2			
13.7-14.6	7.9	2.5	1.3	0.5	4.2			
14.6-15.5	7.9	2.5	1.3	0.5	4.2			
Mean	7.4	2.8	1.1	0.3	1.7			
Std. Dev.	0.4	1.0	0.2	0.2	1.5			

Table 36 Spoil chemistry as a function of time for drill site S-9 in Area B of the Western Energy Company mine near Colstrip, MT.

Spoil Depth(m)	pH	EC mmhos/cm	Pb	Ni	Zn	Sand	Silt	Clay
				ppm			%	
				February, 1978				
0-.9	7.5	3.8	2.3	0.4	1.2	58.6	24	17.4
.9-1.8	7.3	3.7	0.9	0.1	0.4	72.6	16	11.4
1.8-2.7	7.2	6.1	1.2	0.3	1.1	30.6	46	23.4
2.7-3.7	7.3	5.1	1.3	0.3	1.0	56.6	24	19.4
3.7-4.6	7.3	3.8	0.7	0.0	0.6	70.6	18	11.4
4.6-5.5	7.1	6.1	1.1	0.6	1.6	50.6	28	21.4
5.5-6.4	7.4	3.4	2.6	0.7	1.6	64.6	20	15.4
6.4-7.3	7.1	7.6	2.2	0.3	1.1	44.6	28	27.4
7.3-8.2	7.2	6.2	0.9	0.3	1.0	52.6	28	19.4
8.2-9.1	7.1	7.0	0.9	0.2	1.0	36.6	36	27.4
9.1-10.0	7.2	6.7	2.5	0.2	0.9	40.6	34	25.4
10.0-11.0	7.3	6.4	0.7	0.2	0.9	44.6	32	23.4
11.0-11.9	7.3	5.4	1.7	0.7	1.3	52.6	26	21.4
11.9-12.8	7.4	5.8	3.8	0.5	1.4	50.6	28	21.4
12.8-13.7	7.4	6.0	1.5	0.5	1.0	50.6	28	21.4
13.7-14.6	7.5	4.8	2.0	0.8	1.6	54.6	26	19.4
14.6-15.5	7.4	5.2	2.3	0.9	1.3	48.6	30	21.4
15.5-16.5	7.7	2.1	3.1	0.6	0.6	78.6	14	7.4
Mean	7.3	5.3	1.8	0.4	1.1	53.3	27.0	19.7
Std. Dev.	0.2	1.4	0.9	0.3	0.3	12.4	7.6	5.5
				October, 1979				
0-.9	7.4	2.5	3.6	1.1	3.4	40.6	26	33.4
.9-1.8	7.5	2.3	2.8	0.8	2.6	50.6	26	23.4
1.8-2.7	7.5	1.2	1.1	0.1	0.7	72.6	12	15.4
2.7-3.7	7.4	4.4	0.7	0.1	0.4	64.6	16	19.4
3.7-4.6	7.4	3.2	0.9	0.2	0.6	68.6	14	17.4
4.6-5.5	7.4	4.1	0.8	0.3	0.3	62.6	18	19.4
5.5-6.4	7.5	3.8	1.4	0.4	1.0	54.6	22	23.4
6.4-7.3	7.5	3.9	0.9	0.3	0.6	58.6	22	19.4
7.3-8.2	7.5	2.2	1.1	0.4	1.2	52.6	26	21.4
8.2-9.1	7.5	4.1	0.6	0.3	0.4	74.6	8	17.4
9.1-10.0	6.5	3.6	1.4	0.3	1.3	54.6	22	23.4
10.0-11.0	6.8	1.8	1.8	0.7	1.3	66.6	16	17.4
11.0-11.9	6.9	1.7	2.4	1.0	1.4	62.6	18	19.4
11.9-12.8	7.0	1.9	2.1	0.9	1.6	60.6	18	21.4
12.8-13.7	7.1	2.0	1.3	0.3	0.8	72.6	10	17.4
13.7-14.6	7.0	4.0	1.1	0.2	0.5	58.6	20	21.4
14.6-15.5	7.2	3.3	2.4	1.1	1.2	44.6	26	29.4
15.5-16.5	7.2	2.6	7.0	1.6	1.3	16.6	32	51.4
				April, 1979				
0-.9	7.3	4.1	0.9	0.3	2.1			
.9-1.8	7.7	1.7	0.9	0.2	2.8			
1.8-2.7	7.4	2.1	1.1	0.4	1.9			
2.7-3.7	7.9	2.2	3.1	0.9	4.6			
3.7-4.6	7.9	2.2	3.1	0.9	4.6			
4.6-5.5	7.1	2.1	1.4	0.5	1.4			
5.5-6.4	7.7	2.1	1.4	0.5	1.4			
6.4-7.3	7.7	2.1	1.4	0.5	1.4			
7.3-8.2	7.6	1.9	1.1	0.2	0.4			
8.2-9.1	7.9	2.0	0.7	0.1	0.3			
9.1-10.0	7.6	3.1	0.7	0.2	0.3			
10.0-11.0	7.1	5.0	0.8	0.3	1.0			
11.0-11.9	7.2	4.0	1.6	0.7	1.8			
11.9-12.8	7.3	2.6	2.0	1.1	2.3			
12.8-13.7	7.3	3.5	1.7	0.7	2.1			
13.7-14.6	7.2	3.1	1.5	0.6	2.0			
14.6-15.5	7.3	3.4	1.5	0.5	2.9			
15.5-16.5	7.6	3.3	1.4	0.7	2.0			
Mean	7.5	2.8	1.5	0.5	2.0			
Std. Dev.	0.3	0.9	0.7	0.3	1.2			

Table 37. Spoil chemistry as a function of time for drillsite S-10 in Area B of the Western Energy Company mine near Colstrip, MT.

Spoil		EC	Pb	Ni	Zn	Sand	Silt	Clay
Depth(m)	pH	mmhos/cm		ppm			%	
			February, 1978					
0-.9	7.5	4.6	1.2	0.3	0.5	60.6	18	21.4
.9-1.8	7.2	2.6	0.8	0.3	0.3	78.6	16	5.4
1.8-2.7	7.2	5.9	1.0	0.3	0.8	54.6	28	17.4
2.7-3.7	7.5	2.2	1.1	0.3	0.6	76.6	14	9.4
3.7-4.6	7.3	4.5	1.0	0.3	0.7	66.6	20	13.4
4.6-5.5	7.2	5.8	1.2	0.5	1.1	54.6	26	19.4
5.5-6.4	7.2	5.1	1.5	0.4	0.9	60.6	24	15.4
6.4-7.3	7.4	4.0	1.0	0.4	0.5	66.6	20	13.4
7.3-8.2	7.5	3.6	2.0	0.9	1.7	60.6	22	17.4
8.2-9.1	7.5	4.4	3.2	1.2	2.6	50.6	26	23.4
9.1-10.0	7.4	6.0	1.7	0.7	1.5	44.6	32	23.4
10.0-11.0	7.3	6.8	1.2	0.4	1.3	38.6	36	25.4
11.0-11.9	7.3	5.2	4.4	1.0	1.9	46.6	28	25.4
Mean	7.3	4.7	1.6	0.5	1.1	50.4	23.8	17.7
Std. Dev.	0.1	1.3	1.0	0.3	0.7	11.9	6.3	6.2
			October, 1978					
0-.9	6.7	3.4	0.8	0.2	0.4	76.6	10.	13.4
.9-1.8	6.7	4.9	0.9	0.2	0.8	64.6	14	21.4
1.8-2.7	7.0	3.8	0.9	0.1	0.7	44.6	28	27.4
2.7-3.7	6.9	5.0	0.8	0.2	0.4	46.6	14	39.4
3.7-4.6	7.1	4.4	0.8	0.2	0.3	46.6	24	29.4
4.6-5.5	7.2	4.0	0.7	0.2	0.2	58.6	22	19.4
5.5-6.4	7.1	4.2	0.7	0.2	0.5	56.6	24	19.4
6.4-7.3	7.1	4.5	0.9	0.2	0.6	58.6	22	19.4
7.3-8.2	7.2	4.0	0.9	0.4	0.6	64.6	16	19.4
8.2-9.1	7.3	5.7	0.8	0.1	0.4	54.6	26	19.4
9.1-10.0	7.3	4.4	1.0	0.2	0.5	54.6	24	21.4
10.0-11.0	7.3	4.6	1.2	0.3	0.8	72.6	4	23.4
11.0-11.9	7.3	3.5	3.0	1.1	2.2	46.6	28	25.4
Mean	7.1	4.3	1.0	0.3	0.6	57.4	19.7	22.9
Std. Dev.	0.2	0.6	0.6	0.3	0.5	10.2	7.4	6.4
			April, 1979					
0-.9	7.4	4.9	0.8	0.1	5.2			
.9-1.8	7.6	1.8	1.5	0.3	5.8			
2.7-3.7	7.9	1.7	1.3	0.1	1.6			
3.7-4.6	7.3	5.2	1.7	0.6	2.4			
4.6-5.5	7.5	3.3	1.3	0.6	1.8			
5.5-6.4	7.6	2.0	0.9	0.3	1.8			
6.4-7.3	7.4	2.4	1.2	0.4	1.0			
7.3-8.2	7.7	1.5	1.8	0.6	1.8			
8.2-9.1	7.7	2.5	0.9	0.1	0.4			
9.1-10.0	7.2	3.8	0.9	0.1	0.5			
10.0-11.0	7.2	3.8	0.9	0.1	0.5			
11.0-11.9	7.2	3.8	0.9	0.1	0.5			
Mean	7.5	2.9	1.2	0.3	1.9			
Std. Dev.	0.2	1.3	0.3	0.2	1.7			

Table 38. Spoil chemistry as a function of time for drill site S-12 in Area B of the Western Energy Company mine near Colstrip, MT.

Spoil Depth(m)	pH	EC mmhos/cm	Pb	Ni	Zn	Sand	Silt	Clay
				ppm			%	
			February, 1978					
0-.9	7.5	3.3	5.0	2.1	6.3	40.6	24	35.4
.9-1.8	7.4	1.7	1.3	0.4	0.6	74.6	14	11.4
1.8-2.7	7.4	3.0	3.7	1.2	3.5	36.6	28	35.4
2.7-3.7	7.2	3.8	1.3	0.4	0.8	50.6	28	21.4
3.7-4.6	7.2	4.9	1.2	0.5	1.4	52.6	26	21.4
4.6-5.5	7.4	2.9	0.9	0.3	0.7	64.6	20	15.4
5.5-6.4	7.1	5.6	1.4	0.4	1.1	48.6	30	21.4
6.4-7.3	7.0	6.4	1.1	0.3	1.0	38.6	36	25.4
7.3-8.2	7.0	5.0	1.4	0.3	0.8	58.6	24	17.4
8.2-9.1	7.0	6.0	1.1	0.3	0.9	50.6	28	21.4
9.1-10.0	7.4	3.0	0.9	0.2	0.4	60.6	24	15.4
10.0-11.0	7.1	6.0	0.9	0.2	0.7	50.6	26	23.4
11.0-11.9	7.0	5.7	1.5	0.6	1.4	44.6	28	27.4
11.9-12.8	7.3	4.0	2.9	0.6	1.3	54.6	22	23.4
12.8-13.7	7.3	2.9	3.2	0.8	1.8	64.6	20	15.4
13.7-14.6	7.4	2.1	1.1	0.4	0.7	76.6	16	7.4
14.6-15.5	7.7	3.0	2.4	0.8	1.9	58.6	22	19.4
Mean	7.3	4.1	1.8	0.6	1.5	54.5	24.5	21.0
Std. Dev.	0.2	1.5	1.2	0.5	1.4	11.5	5.3	7.4
			October, 1978					
0-.9	7.7	1.2	0.7	0.3	0.1	72.6	14	13.4
.9-1.8	7.5	3.7	0.7	0.3	0.2	64.6	16	19.4
1.8-2.7	7.5	3.3	2.8	0.9	3.4	52.6	20	27.4
2.7-3.7	7.5	3.0	3.5	1.5	3.4	40.6	24	35.4
3.7-4.6	7.5	2.7	1.8	0.6	1.3	56.6	20	23.4
4.6-5.5	7.6	1.9	1.0	0.3	0.4	74.6	10	15.4
5.5-6.4	7.6	1.9	1.2	0.6	0.3	72.6	12	15.4
6.4-7.3	7.6	1.7	1.1	0.5	0.2	74.6	12	13.4
7.3-8.2	7.5	3.2	0.7	0.5	0.4	64.6	16	19.4
8.2-9.1	7.6	4.5	0.7	0.4	0.1	66.6	16	17.4
9.1-10.0	7.4	2.6	0.9	0.4	0.5	60.6	20	19.4
10.0-11.0	7.4	4.0	1.6	0.8	1.5	54.6	22	23.4
11.0-11.9	7.4	3.7	1.6	0.7	1.1	54.6	24	21.4
11.9-12.8	7.5	2.8	1.8	0.9	1.5	54.6	24	21.4
12.8-13.7	7.6	2.5	1.6	0.7	1.2	58.6	20	21.4
13.7-14.6	7.5	3.0	1.3	0.5	0.9	66.6	14	19.4
14.6-15.5	7.4	3.7	1.1	0.4	0.6	60.6	20	19.4
Mean	7.5	2.9	1.4	0.6	1.0	61.8	17.9	20.3
Std. Dev.	0.1	0.9	0.8	0.3	1.0	9.2	4.5	5.3
			April, 1979					
0-.9		4.5	1.2	0.5	2.5			
.9-1.8	7.4	2.7	1.7	0.8	2.7			
1.8-2.7	7.3	1.8	1.1	0.4	0.8			
2.7-3.7	1.3	1.8	1.1	0.4	0.8			
3.7-4.6	7.3	2.7	0.7	0.1	0.3			
4.6-5.5	7.3	2.7	0.7	0.1	0.3			
5.5-6.4	7.3	2.7	0.7	0.1	0.3			
6.4-7.3	7.3	2.7	0.7	0.1	0.3			
7.3-8.2	7.3	2.7	0.7	0.1	0.3			
8.2-9.1	7.3	2.7	0.7	0.1	0.3			
9.1-10.0	7.3	2.7	0.7	0.1	0.3			
10.0-11.0	7.3	3.0	0.9	0.4	1.0			
11.0-11.9	7.3	3.0	0.9	0.4	1.0			
11.9-12.8	7.3	3.0	0.9	0.4	1.0			
12.8-13.7	7.2	3.8	1.6	0.6	1.2			
13.7-14.6	7.4	3.6	1.2	0.4	1.1			
14.6-15.5	7.1	3.6	1.2	0.5	1.1			
Mean	7.3	2.9	1.0	0.4	1.1			
Std. Dev.	0.1	0.7	0.4	0.3	1.0			

Table 39. Spoil chemistry as a function of time for drill stie S-13 in Area B of the Western Energy Company mine near Colstrip, MT.

Spoil Depth(m)	pH	EC mmhos/cm	Pb	Ni	Zn	Sand	Silt	Clay
				ppm			%	
			February, 1978					
0-.9	7.5	1.6	1.5	0.4	0.9	74.6	18	7.4
.9-1.8	7.4	2.2	1.4	0.6	1.5	68.6	18	13.4
1.8-2.7	7.5	1.4	0.5	0.3	0.0	80.6	14	5.4
2.7-3.7	7.4	1.6	0.9	0.3	0.3	72.6	20	7.4
3.7-4.6	7.6	2.4	2.7	0.8	1.6	56.6	18	25.4
4.6-5.5	7.5	3.4	1.7	0.4	1.4	56.6	22	21.4
5.5-6.4	7.6	1.8	1.1	0.4	0.7	72.6	16	11.4
6.4-7.3	7.4	3.4	0.5	0.3	0.2	70.6	18	11.4
7.3-8.2	7.3	3.5	0.6	0.3	0.4	74.6	16	9.4
8.2-9.1	7.2	4.7	1.0	0.4	1.0	58.6	26	15.4
9.1-10.0	7.0	6.8	1.1	0.2	0.6	50.6	28	21.4
10.0-11.0	7.2	5.7	1.7	0.6	1.7	48.6	28	23.4
11.0-11.9	7.4	2.4	1.1	0.4	0.7	76.6	14	9.4
11.9-12.8	7.3	3.2	1.8	0.4	0.5	74.6	16	9.4
12.8-13.7	7.4	2.0	1.4	0.3	0.6	78.6	14	7.4
Mean	7.4	3.1	1.3	0.4	0.8	67.7	19.1	13.3
Std. Dev.	0.2	1.6	0.6	0.2	0.5	10.5	4.8	6.6
			October, 1978					
0-.9	7.5	1.8	1.6	0.4	2.2	74.6	10	15.4
.9-1.8	7.5	1.8	1.1	0.3	0.7	74.6	10	15.4
1.8-2.7	7.5	2.2	0.9	0.2	0.2	72.6	12	15.4
2.7-3.7	7.4	3.4	1.6	0.8	1.8	56.6	22	21.4
3.7-4.6	7.3	4.4	0.7	0.4	0.3	64.6	16	19.4
4.6-5.5	7.2	4.4	1.0	0.4	0.6	50.6	28	21.4
5.5-6.4	7.2	4.1	1.8	0.8	1.7	56.6	22	21.4
6.4-7.3	7.4	1.7	1.4	0.5	0.6	52.6	34	13.4
7.3-8.2	7.5	2.0	1.3	0.4	1.1	64.6	16	19.4
8.2-9.1	7.6	1.6	0.9	0.1	0.2	76.6	10	13.4
9.1-10.0	7.5	2.2	0.9	0.2	0.1	78.6	10	11.4
10.0-11.0	7.5	2.8	0.7	0.3	0.2	64.6	18	17.4
11.0-11.9	7.4	3.8	1.1	0.4	0.8	60.6	20	19.4
11.9-12.8	7.4	4.2	0.8	0.3	0.3	58.6	22	19.4
12.8-13.7	7.3	4.0	1.3	0.7	1.1	56.6	22	21.4
Mean	7.4	3.0	1.1	0.4	0.8	64.2	18.1	17.7
Std. Dev.	0.1	1.1	0.3	0.2	0.7	9.2	7.2	3.4
			April, 1979					
0-.9	7.2	4.0	1.1	0.4	1.5			
.9-1.8	7.3	2.4	2.3	1.6	4.8			
1.8-2.7	7.5	3.7	1.2	0.5	0.9			
2.7-3.7	7.2	2.7	2.4	1.0	2.6			
3.7-4.6	7.3	4.9	0.9	0.3	0.6			
4.6-5.5	7.4	1.4	1.7	0.7	2.2			
5.5-6.4	7.3	3.9	1.1	0.5	3.3			
6.4-7.3	7.8	1.2	1.2	0.3	0.4			
7.3-8.2	7.6	3.6	1.5	0.6	1.4			
8.2-9.1	7.7	1.7	1.0	0.4	0.3			
9.1-10.0	7.1	1.3	1.0	0.3	0.6			
10.0-11.0	7.2	3.0	1.2	0.4	1.1			
11.0-11.9	7.4	2.6	1.3	0.7	1.4			
11.9-12.8	7.1	2.5	1.2	0.5	1.3			
12.8-13.7	7.3	3.4	1.4	0.7	3.1			
Mean	7.4	2.8	1.4	0.6	1.7			
Std. Dev.	0.2	1.1	0.4	0.3	1.3			

Table 40. Spoil chemistry as a function of time for drill site S-15 in Area B of the Western Energy Company mine near Colstrip, MT.

Spoil Depth(m)	pH	EC mmhos/cm	Pb	Ni	Zn	Sand	Silt	Clay
				ppm			%	
				February, 1978				
0-.9	7.5	2.8	2.6	0.7	3.8	52.6	22	25.4
.9-1.8	7.4	7.9	1.0	0.2	1.3	74.6	12	13.4
1.8-2.7	7.4	3.4	4.1	1.6	8.3	36.6	26	39.4
2.7-3.7	7.8	2.1	1.2	0.1	1.0	78.6	14	7.4
3.7-4.6	7.6	2.1	1.5	0.2	1.4	60.6	20	19.4
4.6-5.5	7.5	2.8	3.5	1.1	3.2	50.6	26	23.4
5.5-6.4	7.4	2.5	1.6	0.3	1.2	74.6	14	11.4
6.4-7.3	7.6	2.3	1.5	0.4	1.2	70.6	16	13.4
7.3-8.2	7.4	4.2	0.7	0.0	0.6	66.6	20	13.4
8.2-9.1	7.2	4.1	1.0	0.2	0.9	72.6	16	11.4
9.1-10.0	7.1	5.8	0.7	0.2	0.7	50.6	40	9.4
10.0-11.0	7.3	5.0	1.8	0.6	2.6	58.6	24	17.4
11.0-11.9	7.4	3.2	1.3	0.2	1.1	76.6	16	7.4
11.9-12.8	7.4	3.2	1.7	0.4	1.7	74.6	16	9.4
12.8-13.7	7.4	1.2	0.7	0.2	0.7	86.6	10	3.4
13.7-14.6	7.3	2.6	1.0	0.3	0.9	80.6	14	5.4
14.6-15.5	7.7	2.4	1.9	0.5	1.8	78	14	8
Mean	7.4	3.0	1.6	0.4	1.9	67.3	18.8	14.0
Std. Dev.	0.2	1.2	1.0	0.4	1.9	13.5	7.2	8.9
				October, 1978				
0-.9	7.3	1.6	1.3	0.3	0.9	74.6	12	13.4
.9-1.8	7.2	2.4	1.1	0.2	0.6	76.6	10	13.4
1.8-2.7	7.2	1.6	1.6	0.4	1.3	70.6	12	17.4
2.7-3.7	7.3	1.8	2.7	1.0	2.6	64.6	14	21.4
3.7-4.6	7.4	1.2	1.3	0.6	0.7	76.6	10	13.4
4.6-5.5	7.4	1.6	1.1	0.4	0.5	72.6	14	13.4
5.5-6.4	7.3	2.4	1.0	0.1	0.5	80.6	8	11.4
6.4-7.3	7.3	3.1	0.9	0.2	0.4	68.6	14	17.4
7.3-8.2	7.3	3.2	1.0	0.2	0.4	74.6	10	15.4
8.2-9.1	7.2	3.6	0.9	0.1	0.2	74.6	12	13.4
9.1-10.0	7.2	4.5	1.3	0.4	0.9	54.6	24	21.4
10.0-11.0	7.2	3.9	1.2	0.2	1.0	48.6	26	25.4
11.0-11.9	7.3	2.9	1.2	0.4	1.0	54.6	24	21.4
11.9-12.8	7.2	4.9	1.0	0.1	0.5	52.6	24	23.4
12.8-13.7	7.2	4.0	1.1	0.2	0.5	40.6	30	29.4
13.7-14.6	7.1	4.6	1.2	0.2	0.6	32.6	34	33.4
14.6-15.5	7.2	4.1	1.9	0.8	1.1	50.6	24	25.4
Mean	7.3	3.0	1.3	0.3	0.8	62.8	17.8	19.4
Std. Dev.	0.1	1.2	0.4	0.2	0.5	14.4	8.1	6.4
				April, 1979				
0-.9	7.1	2.0	2.6	0.9	3.2			
.9-1.8	7.1	1.8	1.6	0.7	1.3			
1.8-2.7	7.2	1.2	1.0	0.2	0.2			
2.7-3.7	7.7	0.7	1.0	0.2	0.1			
3.7-4.6	7.2	1.5	1.1	0.2	0.1			
4.6-5.5	7.4	2.1	1.0	0.3	0.4			
5.5-6.4	7.1	2.4	1.5	0.4	5.3			
6.4-7.3	7.4	2.3	1.2	0.4	31.			
7.3-8.2	7.2	4.4	1.2	0.6	7.9			
8.2-9.1	7.4	2.5	1.5	0.7	9.8			
9.1-10.0	7.9	0.7	0.7	0.3	3.0			
10.0-11.0	7.7	1.4	1.6	0.8	3.1			
11.0-11.9	7.6	2.0	1.2	0.8	5.1			
11.9-12.8	7.5	2.8	2.0	0.9	21.			
12.8-13.7	7.6	1.6	1.6	0.5	1.2			
13.7-14.6	7.6	2.9	2.2	0.9	21.			
14.6-15.5	7.7	3.2	2.2	0.8	29.			
Mean	7.4	2.1	1.5	0.6	8.4			
Std. Dev.	0.2	0.9	0.5	0.3	10.4			

Table 41. Spoil chemistry as a function of time for drill site S-16 in Area B of the Western Energy Company mine near Colstrip, MT.

Spoil Depth(m)	pH	EC mmhos/cm	Pb	Ni	Zn	Sand	Silt	Clay
				ppm			%	
				February, 1978				
0-.9	7.5	1.7	1.2	0.2	0.5	84.6	12	3.4
.9-1.8	7.3	3.7	0.8	0.2	0.6	74.6	18	7.4
1.8-2.7	7.1	6.1	1.1	0.3	1.6	56.6	28	15.4
2.7-3.7	7.0	5.4	0.8	0.1	0.8	64.6	22	13.4
3.7-4.6	7.0	6.6	1.2	0.3	1.0	52.6	26	21.4
4.6-5.5	7.0	3.3	2.2	0.8	2.8	66.6	18	15.4
5.5-6.4	7.2	2.9	2.0	0.8	2.1	64.6	20	15.4
6.4-7.3	7.4	2.9	4.1	1.5	4.3	38.6	30	31.4
7.3-8.2	7.4	2.0	2.4	0.8	2.2	44.6	38	17.4
8.2-9.1	7.5	1.2	0.8	0.1	0.3	82.6	12	5.4
9.1-10.0	7.1	5.2	0.5	0.1	0.3	71.6	16	9.4
10.0-11.0	6.9	5.2	0.7	0.0	0.5	62.6	26	11.4
11.0-11.9	6.1	5.3	0.6	0.0	0.3	68.6	20	11.4
11.9-12.8	7.2	2.2	0.6	0.1	0.5	48.6	32	19.4
12.8-13.7	7.0	5.3	2.2	0.3	1.4	60.6	24	15.4
13.7-14.6	7.3	5.3	0.7	0.1	0.5	58.6	28	13.4
14.6-15.5	7.3	3.8	0.6	0.1	0.5	66.6	22	11.4
15.5-16.5	7.1	4.7	0.6	0.0	0.5	56.6	30	13.4
16.5-17.4	7.2	1.7	1.6	0.7	2.1	78.6	12	9.4
Mean	7.2	3.9	1.3	0.3	1.2	63.3	22.8	13.7
Std. Dev.	0.2	1.7	0.9	0.4	1.1	12.3	7.3	6.2
				October, 1978				
0-.9	7.4	2.1	1.7	0.5	2.4	76.6	10	13.4
.9-1.8	7.4	1.7	1.3	0.6	1.1	74.6	12	13.4
1.8-2.7	7.3	3.6	1.4	0.5	1.0	60.6	18	21.4
2.7-3.7	7.3	3.7	1.8	0.7	1.0	66.6	16	17.4
3.7-4.6	7.3	3.4	2.0	0.8	1.1	64.6	16	19.4
4.6-5.5	6.6	2.0	2.2	0.9	1.8	58.6	18	23.4
5.5-6.4	6.5	1.8	1.3	0.3	0.7	74.6	12	13.4
6.4-7.3	6.8	1.3	0.9	0.1	0.2	82.6	6	11.4
7.3-8.2	6.8	2.2	1.0	0.1	0.4	64.6	18	17.4
8.2-9.1	6.9	2.8	0.9	0.3	0.5	68.6	14	17.4
9.1-10.0	7.0	4.7	0.8	0.2	0.5	76.6	10	13.4
10.0-11.0	7.0	3.6	1.1	0.2	0.8	58.6	22	19.4
11.0-11.9	7.0	3.4	1.4	0.4	1.4	54.6	24	21.4
11.9-12.8	7.0	3.8	1.4	0.5	1.7	48.6	28	23.4
12.8-13.7	7.1	3.2	1.8	0.6	1.6	56.6	24	19.4
13.7-14.6	7.3	1.5	1.0	0.2	0.4	72.6	12	15.4
14.6-15.5	7.3	2.8	1.1	0.3	0.6	72.6	12	15.4
15.5-16.5	7.3	2.9	1.3	0.5	1.3	64.6	18	17.4
16.5-17.4	7.3	2.7	1.3	0.6	1.1	54.6	24	21.4
Mean	7.1	2.8	1.4	0.4	1.0	65.9	16.5	17.6
Std. Dev.	0.3	0.9	0.4	0.2	0.6	9.2	5.9	3.6
				April, 1979				
0-.9	7.3	2.7	2.0	0.9	2.3			
.9-1.8	7.6	2.7	1.4	0.8	1.0			
1.8-2.7	7.4	1.2	1.6	0.4	1.4			
2.7-3.7	7.3	1.0	1.3	0.5	1.0			
3.7-4.6	7.4	1.4	1.4	0.4	1.3			
4.6-5.5	7.6	4.7	1.4	0.2	1.0			
5.5-6.4	7.3	3.5	1.5	0.4	1.9			
6.4-7.3	6.9	1.9	2.6	1.7	5.4			
7.3-8.2	7.1	1.5	3.1	0.7	3.9			
8.2-9.1	7.0	1.4	2.2	0.8	2.3			
9.1-10.0	7.0	1.4	1.1	0.2	0.3			
10.0-11.0	7.0	2.2	1.0	0.2	0.5			
11.0-11.9	7.2	1.3	1.5	0.6	2.5			
11.9-12.8	7.2	1.3	1.5	016	2.5			
12.8-13.7	7.3	0.7	1.0	0.1	2.6			
13.7-14.6	7.1	1.1	1.2	0.4	1.6			
14.6-15.5	7.2	2.5	1.1	0.4	1.5			
15.5-16.5	7.0	3.3	0.9	0.2	10.2			
16.5-17.4	7.0	3.7	1.6	0.5	3.6			
Mean	7.2	2.1	1.5	0.5	2.5			
Std. Dev.	0.2	1.1	0.6	0.4	2.3			

Table 42. Spoil chemistry as a function of time for drill site S-18 in Area B of the Western Energy Company mine near Colstrip, MT.

Spoil Depth(m)	pH	EC mmhos/cm	Pb	Ni	Zn	Sand	Silt	Clay
				ppm			%	
February, 1978								
0-.9	7.5	1.2	0.9	0.2	0.7	82.6	12	5.4
.9-1.8	7.5	1.2	1.0	0.0	0.4	84.6	12	3.4
1.8-2.7	7.5	1.3	0.8	0.0	0.3	86.6	10	3.4
2.7-3.7	7.1	4.5	0.7	0.0	0.3	76.6	18	5.4
3.7-4.6	7.3	2.6	0.7	0.1	0.3	80.6	12	7.4
4.6-5.5	7.2	3.1	0.9	0.2	0.5	76.6	14	9.4
5.5-6.4	7.1	3.9	1.0	0.2	0.4	76.6	14	9.4
6.4-7.3	7.3	3.3	0.8	0.1	0.4	76.6	14	9.4
7.3-8.2	7.3	3.4	0.5	0.0	0.2	80.6	12	7.4
8.2-9.1	7.3	3.5	0.4	0.1	0.5	70.6	18	11.4
9.1-10.0	7.1	5.1	0.4	0.1	0.4	76.6	14	9.4
10.0-11.0	7.2	4.3	0.3	0.1	0.3	58.6	26	15.4
11.0-11.9	7.0	5.4	0.5	0.1	0.5	60.6	24	15.4
11.9-12.8	7.2	4.0	1.1	0.0	0.5	74.6	16	9.4
12.8-13.7	7.2	4.1	2.1	0.0	0.5	78.6	14	7.4
13.7-14.6	7.2	3.8	0.7	0.0	0.6	60.6	24	15.4
14.6-15.5	7.3	5.2	0.6	0.2	0.4	58.6	28	13.4
15.5-16.5	6.1	5.5	0.8	0.1	0.4	50.6	34	15.4
16.5-17.4	6.8	6.0	1.1	0.1	0.5	50.6	32	17.4
17.4-18.3	7.3	2.7	1.8	0.5	1.5	70.6	16	13.4
Mean	7.2	3.7	0.9	0.1	0.5	71.6	18.2	10.2
Std. Dev.	0.3	1.4	0.4	0.1	0.3	11.1	7.2	4.3
October, 1978								
0-.9	7.4	1.9	2.2	0.7	2.7	68.6	14	17.4
.9-1.8	7.5	1.7	3.5	1.0	2.7	54.6	18	27.4
1.8-2.7	7.5	1.7	1.5	0.4	1.1	76.6	12	11.4
2.7-3.7	7.5	1.2	1.1	0.1	0.3	78.6	10	11.4
3.7-4.6	7.6	0.9	1.1	0.2	0.3	82.6	6	11.4
4.6-5.5	7.5	1.9	1.1	0.1	1.5	84.6	4	11.4
5.5-6.4	7.4	2.1	1.3	0.2	0.7	80.6	8	11.4
6.4-7.3	7.3	3.2	1.1	0.3	0.4	74.6	12	13.4
7.3-8.2	7.4	3.4	1.1	0.2	0.3	74.6	12	13.4
8.2-9.1	7.4	4.0	1.6	0.4	1.1	60.6	20	19.4
9.1-10.0	7.5	1.2	1.5	0.5	0.9	48.6	28	23.4
10.0-11.0	7.4	2.1	1.6	0.5	1.0	48.6	26	25.4
11.0-11.9	6.8	1.5	1.6	0.6	1.3	48.6	26	25.4
11.9-12.8	6.8	2.2	1.3	0.4	0.9	48.6	26	25.4
12.8-13.7	7.1	2.2	2.3	1.1	2.4	50.6	22	27.4
13.7-14.6	7.1	1.6	1.9	0.8	1.3	68.6	16	15.4
14.6-15.5	7.0	3.2	1.7	0.6	1.1	56.6	22	21.4
15.5-16.5	7.2	1.8	1.4	0.3	0.6	68.6	14	17.4
16.5-17.4	7.3	2.0	2.1	0.6	1.7	64.6	14	21.4
17.4-18.3	7.3	2.3	1.9	0.5	1.2	62.6	18	19.4
Mean	7.3	2.1	1.6	0.5	1.2	65.1	16.4	18.5
Std. Dev.	0.2	0.8	0.6	0.3	0.7	12.5	7.0	5.9
April, 1979								
0-.9	7.2	1.1	1.1	0.1	0.3			
.9-1.8	7.2	1.9	1.0	0.0	0.5			
1.8-2.7	7.4	1.1	1.2	0.1	0.4			
2.7-3.7	7.1	2.0	1.0	0.2	0.7			
3.7-4.6	7.1	1.6	1.1	0.1	0.1			
4.6-5.5	7.3	0.9	1.1	0.0	0.5			
5.5-6.4	7.1	1.9	1.3	0.2	2.1			
6.4-7.3	7.3	1.2	1.5	0.3	1.9			
7.3-8.2	7.5	0.6	0.7	0.0	0.6			
8.2-9.1	7.3	0.7	0.7	0.1	0.3			
9.1-10.0	7.2	1.8	0.7	0.1	0.5			
10.0-11.0	7.1	3.3	0.7	0.1	0.3			
11.0-11.9	7.4	0.9	0.8	0.2	0.5			
11.9-12.8	7.0	3.9	0.7	0.2	1.7			
12.8-13.7	7.0	4.6	0.8	0.2	1.3			
13.7-14.6	7.1	4.3	0.6	0.2	1.1			
14.6-15.5	7.1	3.0	1.4	0.3	2.5			
15.5-16.5								
16.5-17.4	7.1	1.7	1.1	0.2	1.5			
17.4-18.3	7.1	1.8	1.0	0.1	2.8			
Mean	7.2	2.0	1.0	0.1	1.0			
Std. Dev.	0.1	1.2	0.3	0.1	0.8			

Table 43. Spoil chemistry as a function of time for drill site S-19 in Area B of the Western Energy Company mine near Colstrip, MT.

Spoil		EC	Pb	Ni	Zn	Sand	Silt	Clay
Depth(m)	pH	mmhos/cm		ppm			%	
February, 1978								
0-.9	7.6	1.2	0.9	0.0	0.2	82.6	12	5.4
.9-1.8	7.4	2.2	0.8	0.0	0.2	84.6	10	5.4
1.8-2.7	7.4	2.4	1.0	0.3	0.8	80.6	14	5.4
2.7-3.7	7.5	1.8	1.1	0.3	0.8	82.6	12	5.4
3.7-4.6	7.5	2.0	0.9	0.0	0.3	82.6	12	5.4
4.6-5.5	7.4	3.4	0.7	0.0	0.2	82.6	12	5.4
5.5-6.4	7.4	3.6	0.8	0.1	0.3	82.6	12	5.4
6.4-7.3	7.5	2.8	0.7	0.0	0.2	84.6	10	5.4
7.3-8.2	7.3	4.3	0.6	0.0	0.2	78.6	14	7.4
8.2-9.1	7.0	6.0	0.7	0.1	0.4	70.6	20	9.4
9.1-10.0	7.0	4.8	0.6	0.1	0.2	76.6	16	7.4
10.0-11.0	7.3	5.8	0.6	0.2	0.4	60.6	26	13.4
11.0-11.9	7.3	3.6	0.7	0.1	0.3	68.6	20	11.4
11.9-12.8	7.6	3.2	1.2	0.1	0.2	84.6	10	5.4
12.8-13.7	7.6	2.7	0.7	0.0	0.2	88.6	8	3.4
13.7-14.6	7.1	5.3	0.7	0.0	0.4	76.6	16	7.4
14.6-15.5	7.2	5.4	2.0	0.2	0.6	66.6	22	11.4
15.5-16.5	7.2	5.8	1.4	0.3	1.1	64.6	24	11.4
16.5-17.4	7.4	2.6	1.1	0.2	0.6	84.6	10	5.4
17.4-18.3	7.4	2.9	1.0	0.2	0.4	82.6	12	5.4
Mean	7.4	3.6	0.9	0.1	0.4	78.3	14.6	7.1
Std. Dev.	0.2	1.5	0.3	0.1	0.3	7.9	5.2	2.8
October, 1978								
0-.9	7.4	2.1	1.3	0.2	0.4	72.6	20	7.4
.9-1.8	7.5	1.1	1.1	0.2	0.3	82.6	8	9.4
1.8-2.7	7.5	1.2	1.2	0.1	0.2	80.6	8	11.4
2.7-3.7	7.4	2.1	1.4	0.2	0.8	62.6	18	19.4
3.7-4.6	7.5	3.6	1.0	0.2	0.2	78.6	10	11.4
4.6-5.5	7.5	3.7	1.6	0.3	0.9	72.6	14	13.4
5.5-6.4	7.5	2.7	2.4	0.7	2.7	60.6	22	17.4
6.4-7.3	7.6	1.8	1.8	0.6	1.1	74.6	12	13.4
7.3-8.2	7.6	2.0	2.4	0.5	2.1	70.6	12	17.4
8.2-9.1	7.6	1.6	0.9	0.5	0.1	80.6	10	9.4
9.1-10.0	7.6	2.3	1.0	0.3	0.4	78.6	10	11.4
10.0-11.0	7.7	1.0	1.0	0.3	0.4	66.6	12	21.4
11.0-11.9	7.6	1.9	2.1	1.0	2.2	52.6	20	27.4
11.9-12.8	7.8	0.4	1.2	0.5	0.7	58.6	16	25.4
12.8-13.7	7.8	0.8	0.8	0.3	0.3	70.6	8	21.4
13.7-14.6	7.8	1.0	0.8	0.3	0.4	72.6	6	21.4
14.6-15.5	7.7	2.0	0.9	0.3	0.2	60.6	22	17.4
15.5-16.5	7.5	3.1	1.0	0.4	0.6	56.6	20	23.4
16.5-17.4	7.5	4.0	0.9	0.2	0.4	56.6	18	25.4
17.4-18.3	7.5	3.6	0.8	0.3	0.4	58.6	18	23.4
Mean	7.6	2.1	1.3	0.4	0.7	68.4	14.2	17.4
Std. Dev.	0.1	1.1	0.5	0.2	0.7	9.4	5.2	6.2
April, 1979								
0-.9	7.2	1.4	2.1	0.6	35.			
.9-1.8	7.0	2.2	1.4	0.4	1.1			
1.8-2.7	7.0	1.9	1.3	0.3	0.5			
2.7-3.7	7.1	1.7	1.7	0.6	1.5			
3.7-4.6	7.0	1.7	1.3	0.4	0.5			
4.6-5.5	7.0	2.0	1.3	0.2	1.6			
5.5-6.4	7.0	1.4	1.5	0.3	2.2			
6.4-7.3	7.1	1.3	1.4	0.3	3.0			
7.3-8.2	7.1	1.3	1.1	0.1	0.7			
8.2-9.1	7.7	0.9	1.0	0.1	0.5			
9.1-10.0	7.1	0.7	1.1	0.1	0.5			
10.0-11.0	7.1	0.8	1.2	0.2	0.3			
11.0-11.9	7.0	0.7	1.1	0.1	0.3			
11.9-12.8	6.9	0.7	1.1	0.0	1.4			
12.8-13.7	7.0	0.8	1.3	0.2	5.4			
13.7-14.6	7.0	1.5	1.3	0.2	2.0			
14.6-15.5	7.0	2.9	1.2	0.2	1.2			
15.5-16.5	7.2	2.2	1.9	0.4	1.8			
16.5-17.4	7.0	2.8	1.4	0.3	1.5			
17.4-18.3	7.0	3.7	1.3	0.3	1.9			
Mean	7.1	1.6	1.4	0.3	3.1			
Std. Dev.	0.2	0.8	0.3	0.2	7.6			

APPENDIX E

NEUTRON PROBE CALIBRATION PROCEDURE

Researchers have shown that site specific calibrations should be constructed for the neutron probe method in order to minimize error introduced by different soil types (Shirazi and Isobe, 1976; Rawls and Asmussen, 1973). Air-ejected cuttings from drilling of the access hold for the neutron probe were composited into 90 cm increments. These samples were placed in plastic bags to prevent moisture loss, and taken to a field laboratory. In the lab a representative subsample from each depth increment was weighed, dried at 105°C, and then weighed again to determine the oven-dry gravimetirc water content. The wet bulk density (D_{bw}) of each sample depth was determined *in situ* by gamma-ray attenuation methods. After converting the D_{bw} to a dry bulk density (D_b), the water content by volume of each sample was calculated (Equation 1).

$$\text{Water content, \% by Vol.} = (\text{Water Content, \% by Weight})(D_b). \qquad \text{Eq. 1}$$

In situ spoil water readings were taken with the neutron probe the same day that the samples were obtained. A linear regression analysis was performed on the count ratios obtained from the neutron probe and on the corresponding volumetric water content from each depth. Since the D_{bw} measurement introduced a slight error into the data, a mathematical iteration was also performed to minimize this error. The results of these analyses appear in Figure 3.

After installation of the second series of neutron access tubes in April, 1979, it became necessary to use a 3.8 cm diameter neutron probe since some aluminum access tube joints obstructed the standard 5 cm diameter probe. A calibration equation for the smaller diameter probe was determined by taking neutron probe readings with both probes in five separate access tubes which were installed in mine spoil materials. A linear regression analysis was performed on the neutron probe count ratio from the 3.8 cm diameter probe versus the corresponding volumetric water content % determined by the 5 cm diameter probe.

XIV

LITERATURE CITED

Abeele, W.V. 1978. The influence of access hole parameters on neutron moisture probe readings. U.S. Department of Energy Report LA-7241-MS, Washington, D.C., 5 p.

Allen, J.R.L. 1965. A review of the origin and characteristics of recent alluvial sediments. Sedimentology, Vol. 5, No. 2, pp. 89-191.

Arnold, F.B. and D.J. Dollhopf. 1977. Soil water and solute movement in Montana strip mine spoils. Mt. Agr. Exp. Sta. Rep. 106. Mt. State Univ., Bozeman.

Beaumont, E.A. 1979. Depositional environments of Fort Union sediments (Tertiary, Northwest Colorado) and their relation to coal. AAPG Bull. V-63, No. 2 pp. 194-217.

Bruhn, R.W., V.F. Pierce and R.E. Gray. 1980. An assessment of the feasibility of developing a toxic overburden detection system for surface coal mine applications in the western United States. U.S. Bureau of Mines Final Rep. on Contract J0188051.

Carmichael, V.W. and Burlington Northern Staff. 1964. Photogeologic map of the Colstrip coal field, in Matson, R.E. and Blumer, J.W., 1974. Quality and reserves of strippable coal, selected deposits, southeastern Montana: Mt. Bur. Mines and Geol. Bull: 91, 1974.

Chadwick, R.A., R.A. Woodriff, R.W. Stone and C.M. Bennett. 1975. Lateral and vertical variations in sulfur and trace elements in coal; Colstrip field, Montana. in Proceedings of the Fort Union Coal Field Symposium, V. 3, Mt. Acad. of Sci., Eastern Mt. College, Billings, Mt.

Cooper, H.H., Jr., J.D. Bredehoeff, I.S. Papadopulos. 1967. Response of a finite-diameter well to an instantaneous charge of water. Water Resource Research V3, no. 1. p. 263-269.

Davis, S.N. and R.J.M. DeWiest. 1967. Hydrogeology. John Wiley and Sons, Inc. New York, London, Sidney. 463 pp.

Day, P.R. 1965. Particle fractionation and particle size analysis. in Methods of Soil Analysis. Am. Soc. Agron. Monography No. 9. Madison, Wis., p. 545-566.

DePuit, E.J., W.H. Willmuth and J.G. Coenenberg. 1975. Vegetation and watershed analysis, Tanner Creek study area. Unpub. Mt. Agr. Exp. Sta. Prog. Rep. for Shell Oil Co., Mt. State Univ., Bozeman. 83 p.

Dollhopf, D.J., I.B. Jensen and R.L. Hodder. 1977a. Effects of surface configuration in water pollution control on semiarid mined lands, Mont. Agr. Exp. Sta. Research Report 114, Mt. State Univ., Bozeman.

Dollhopf, D.J., W.D. Hall, C.A. Cull and R.L. Hodder. 1977b. Selective placement of coal strip mine overburden in Montana, II. Initial field demonstration. Mt. Agr. Exp. Sta. Res. Rep. 125. Mt. State Univ., Bozeman. NTIS PB81-103178.

Dollhopf, D.J., W.D. Hall, W.M. Schafer, E.J. DePuit and R.L. Hodder. 1978a. Selective placement of coal strip mine overburden in Montana, I. Data base. Mt. Agr. Exp. Sta. Res. Rep. 128, Mt. State Univ., Bozeman. NTIS PB81-103186.

Dollhopf, D.J., J.D. Goering, C.J. Levine, B.J. Bauman, D.W. Hedberg and R.L. Hodder. 1978b. Selective placement of coal strip mine overburden in Montana, III. Spoil mixing phenomena. Mt. Agr. Exp. Sta. Res. Rep. 135, Mt. State Univ., Bozeman. NTIS PB81-103160.

Dollhopf, D.J., J.D. Goering, C.J. Levine, B.J. Bauman and R.L. Hodder. 1979. Selective placement of coal strip mine overburden in Montana, IV. Hydrogeologic studies. Mt. Agr. Exp. Sta. Res. Rep. 154, Mt. State Univ., Bozeman. NTIS PB81-103194.

Ecological Consulting Service. 1975. Vegetation production, condition, and community typing on Western Energy mining areas A, B, and E. Final Rep. prepared for Western Energy Co., E.C.S. Helena, Mt. 39 p.

Folk, R.L. 1974. Petrology of sedimentary rocks. Hemphill's Bookstore, Austin, Texas. 170 pp.

Gardner, W.R. 1965. Soil water movement and root absorption. in Plant Environment and Efficient Water Use (ed. W.H. Pierre, et al.) Amer. Soc. of Agron. Madison, Wis.

Glaze, R.E. and E.R. Keller. 1965. Geologic History of Powder River Basin. Am. Assoc. Petroleum Geologists Bull., V. 49., No. 11. pp. 1893-1907.

Hinkley, T.K., R.J. Ebens and J.G. Boerngen. 1978. Overburden chemistry and mineralogy at Hanging Woman Creek, Big Horn County, Montana and recommendations for sampling at similar sites. U.S. Dept. of Interior Geological Survey Open-file Report 78-393.

Hounslow, A.W., J.J. Fitzpatrick, L.A. Cerrillo and M. Freland. 1978. Mineralogy of overburden as related to ground-water chemical changes in the strip mining of coal, U.S. EPA-500/7-78-156, Ada, Oklahoma. 299 pp.

Howard, A.D. 1960. Cenozoic history of northeastern Montana and northwestern North Dakota with emphasis on the Pleistocene: U.S. Geol. Survey Prof. Paper 326, 107 p.

Jacob, C.E. 1940. On the flow of water in an elastic artesian aquifer, Am. Geophys. Union Trans., V. 21, pt. 2, p. 574-586.

Johnson Division, Universal Oil Products Co. 1975. Ground Water and Wells. Johnson Division, Universal Oil Products Co., St. Paul, MN, 440 pages.

Krumbine, W.C. and F.J. Pettijohn. 1938. Manual of sedimentary petrography. D. Appleton-Century Co., Inc. New York.

Laurenroth, W.K., J.L. Dodd, R.K. Heituchmidt and R.G. Woodmansee. 1976. Biomass dynamics and primary production in mixed prairie grasslands in southeastern Montana: baseline data for air pollution studies. Appendix A in the bioenvironmental impact of a coal-fired power plant, second interim report, Colstrip, Montana, June 1975 (ed. Lewis, R.A., N.R. Glass, and A.S. Lefohn), EPA-600/3-76-013, Corvallis, Oreg., 314 p.

Lewis, R.A., N.R. Glass and A.S. Lefohn (eds.). 1976. The bioenvironmental impact of a coal-fired power plant. Second interim report, Colstrip, Montana, June, 1975. EPA-600/3-76-013, Corvallis, Oregon. 314 p.

Lewis, R.A. and A.S. Lefohn (eds.). 1976. The bioenvironmental impact of a coal-fired power plant. First interim report, Colstrip, Montana, December, 1974. EPA-600/3-76-002, Corvallis, Oregon. 109 p.

Lindsay, W.L. and W.A. Norvell. 1978. Development of a DTPA soil test for zinc, iron, manganese and copper. Soil Sci. Amer. Jour. Vol. 42. No. 3, pp. 421-28.

Lindsay, W.L. and W.A. Norvell. 1969. Development of a DTPA micronutrient soil test. Agron. Abstr. p. 84. Paper presented at Agron. Soc. Amer. Annual Meeting.

Lohman. S.W. 1972. Ground-water hydraulics. U.S. Geol. Survey Prof. Paper 708.

Mason, B. and L.G. Berry. 1968. Elements of Mineralogy. W.H. Freeman and Co. San Francisco. 550 pp.

McBride, E.F. 1963. A classification of common sandstones. Jour. Sed. Petrology, 33, pp. 664-669.

McHenery, J.R. 1963. Theory and application of neutron scattering in the measurement of soil moisture. Soil Sci. 95:294-307.

McKee, J. and H. Wolf. 1963. Water quality criteria. California State Water Resources Control Board Publication 3-A. Sacramento. 547 pp.

Meyn, R.L., E.S. Sunberg and R.P. Young. 1976. Research on reclamation of surface mined lands at Colstrip, Montana. Mt. Agr. Exp. Sta. Res. Rep. 101, Mt. State Univ., Bozeman. 84 p.

Montana State Department of Natural Resources and Conservation. 1974. Draft environmental impact statement on Colstrip electric generating units 3 & 4, 500 kilovolt transmission lines and associated facilities. Energy Planning Div., Mt. State Dept. of Natural Resources and Conservation, Helena, Montana.

Montana Dept. of State Lands. 1977. Suspect levels of soil parameters. Helena, Mt.

Moran, S.R., G.H. Groenwold and J.A. Cherry. 1978. Hydrogeochemical evolution of soil water and ground water. in Geologic, Hydrologic, and Geochemical Concepts and Techniques in Overburden Characterization of Mined-Land Reclamation. North Dakota Geol. Sur., Rep. 63, pp. 68-82.

Munshower, F.F. and E.J. DePuit. 1976. The effects of stack emission on the range resource in the vicinity of Colstrip, Montana: Progress Rep. 1975. Mt. Agr. Exp. Sta. Res. Rep. 98, Mt. State Univ., Bozeman.

Munshower, F.F., B.W. Sindelar and D.R. Neuman. 1975. The effects of stack emissions on the range resource in the vicinity of Colstrip, MT: Progress Rep. 1972-1974. Mt. Agr. Exp. Sta. Res. Rep. 71, Mt. State Univ., Bozeman, 77 p.

National Academy of Sciences and National Academy of Engineering. 1972. Water quality criteria. U.S. Government Printing Office. Washington, D.C. 594 pp.

Norvell, W.A. and W.L. Lindsay. 1972. Reactions of DTPA chelates of iron, zinc, copper, and manganese with soils. Soil Sci. Soc. Amer. Proc. 36:778-783.

Pagenkopf, G.K. 1978. Introduction to Natural Water Chemistry. Enivronmental Science and Technology Series; V. 3. Marcel Dekker, Inc. New York, New York. 272 pp.

Payne, G.F. 1973. Vegetative rangeland types in Montana. Mt. Agr. Exp. Sta. Bull. 671, Mt. State Univ., Bozeman. 15 p.

Piper, A.M. 1944. A graphic procedure in the geochemical interpretation of water analyses. Am. Geophys. Union Trans., 25:914-923.

Powers, J.F. and F.M. Sandoval. 1976. Effect of sampling method on results of chemical analysis of overburden samples. Mining Congress Journal. pp. 43-47, October.

Rahn, P.H. 1976. Potential of coal stripmine spoils as aquifers in the Powder River Basin. Old West Regional Commission. South Dakota School of Mines and Technology, Rapid City, South Dakota. 108 pp.

Rawls, W.J. and L.E. Asmussen. 1973. Neutron probe field calibration for soils in the Georgia coastal plain. Soil Sci. 116:262-265.

Reineck, Hans-Erich and Indra Bir Singh. 1973. Depositional Sedimentary Environments. Springer-Verlag, Berlin - Heidelberg.

Royse, C.F. Jr. 1970. A sedimentologic analysis of the Tongue River-Sentinel Butte Interval (Paleocene) of the Williston Basin, Western North Dakota. Sedimentary Geology, Vol. 4, pp. 19-80.

Schafer, W.M., G.A. Nielsen, D.J. Dollhopf and K. Temple. 1979. Soil genesis, hydrological properties, root characteristics and microbial activity of 1- to 50-year-old stripmine spoils. U.S. EPA-600/7-79-100, Cincinnati, Ohio. 212 pp.

Schultz, J.D. 1966. Current status of soil moisture measurement by the neutron method in International Symposium on Forest Hydrology, Pergamon Press-Oxford and New York.

Schwaller, R.L. 1981. Personal communication. Professor of Mathematics, Mont. State Univ., Bozeman.

Shirazi, G.A. and M. Isobe. 1976. Calibration of neutron probe in some selected Hawaiian soils. Soil Sci. 122:165-170.

Shurr, G.W. 1972. Paleocurrent indicators in Tongue River sandstones of the Bull Mountain syncline, Montana. in Montana Geol. Society, 21st Ann. Geol. Conf., p. 107-111.

Sindelar, B.W., R.L. Hodder and M.E. Majerus. 1973. Surface mined land reclamation research in Montana. Mt. Agr. Exp. Sta. Res. Rep 40, Mt. State. Univ., Bozeman. 122 p.

Sindelar, B.W., R. Atkinson, M.E. Majerus and K. Proctor. 1972. Surface mined land reclamation research at Colstrip, Montana. Mt. Agr. Exp. Sta. Res. Rep. 69, Mt. State Univ., Bozeman. 98 p.

Skibitzke, H.E. 1958. An equation for potential distribution about a well being bailed: U.S. Geol. Survey Open-File Rept., Ground Water Note 35.

Soil Survey Staff. 1951. Soil survey manual. USDA Handbook No. 18.

Soltanpour, P.N., A. Khan and W.L. Lindsay. 1976. Factors affecting DTPA extractable Zn, Fe, Mn and Cu from soils. Commun. In Soil Science and Plant Analysis 7(9), 797-821.

Sommers, L.E. and W.L. Lindsay. 1979. Effect of pH and redox on predicted heavy metal-chelate equilibria in soils. Soil Sci. Soc. Am. J., Vol. 43:39-47.

Theis, C.V. 1935. The relation between the lowering of the piezometric surface and the rate and duration of discharge of a well using ground-water storage. Am. Geophys. Union Trans., prt. 2.

Toy, T.J. and B.E. Munson. 1978. Climate appraisal maps of rehabilitation potential of strippable coal lands in the Powder River Bain, Wyoming and Montana. U.S. Geol. Survey Miscellaneous Field Study Maps MF-932.

U.S. Dept. of Agriculture. 1954 and 1969. Diagnosis and improvement of saline and alkali soils. Agriculture Handbook No. 60.

U.S. Department of Commerce, NOAA. Climatological summary for Colstrip, MT.

U.S. Environmental Protection Agency. 1974. Methods for chemical analysis of water and wastes. National Environmental Research Center. Cincinnati, Ohio. EPA-625/6-74-003.

U.S. Environmental Protection Agency. 1976. Quality criteria for water. Office of Water Planning Standards. NTIS-PB-263 943.

U.S. Department of Health, Education and Welfare. 1962. Public Health Service drinking water standards. U.S. Public Health Service Publication 956.

U.S. Dept. Interior, Office of Surface Mining. 1977. Surface mining reclamation and enforcement provisions. Title 30, Chapter 7.

Van Voast, W.A. and R.B. Hedges. 1976. Hydrogeologic conditions and projections related to mining near Colstrip, southeastern Montana. Mont. Bur. of Mines and Geology open-file report, June, 1976.

Van Voast, W.A., R.B. Hedges and J.J. McDermott. 1977. Hydrogeologic conditions and projections related to mining near Colstrip, southeastern Montana. Mt. Bur. of Mines and Geol., Bulletin 102.

Van Voast, W.A., R.B. Hedges and J.J. McDermott. 1978. Strip coal mining and mined-land reclamation in the hydrologic system, southeastern Montana. Old West Regional Commission Grant No. 10570165, Billings, Mont., 81 p.

Wentworth, C.K. 1922. A scale of grade and class terms for clastic sediments: Jour. Geol, v. 30, p. 377-392.

Widmayer, M.A. 1977. Depositional model of the sandstone beds in the Tongue River Member of the Fort Union Formation (Paleocene), Decker, MT. unpublished MS thesis, Montana State University.